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PRACTICAL CONSIDERATIONS
of
PYRITE OXIDATION CONTROL
IN URANIUM TAILINGS

A Report for the

NATIONAL URANIUM TAILINGS PROGRAM

By K. D. Hester & Associates
Oakville, Ontario
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July 5, 1984.

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RECEIVED
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Ans'd.

Attention: Mr. D. G. Feasby,
Project Manager.

Dear Mr. Feasby,

Six copies of this report on "Practical Considerations of Pyrite Oxidation Control in Uranium Tailings" is submitted herewith in fulfillment of a Department of Supply and Services Contract, Serial Number 0SQ83-00278, DSS File Number 14SQ.23241-3-1668, issued February 8, 1984 to K. D. Hester & Associates. The contract consignee was the National Uranium Tailings Program, and Mr. D. G. Feasby was the Scientific Authority. The Statement of Work to be covered by the contract is given in Appendix A herein. A preliminary and final draft of the report have previously been reviewed, and the suggested modifications are incorporated in this presentation.

The problems posed by the oxidation of pyrite in uranium tailings includes the generation of sulphuric acid, and acid sulphate metal salts, particularly iron salts. At the concentration levels generated, these have substantial negative impact on the normal biota in watercourses by themselves. The reduced pH levels tend to mobilize heavy metals present in the tailings, and also the radionuclides. These additional metal salts increase the undesirable effects. The rate of oxidation of pyrite at lower pH levels is catalyzed by ubiquitous sulphur and iron oxidizing bacteria present in soils; the bacterially activated catalysis is of major importance in maintaining and increasing the acidity of the pertinent tailings.

No single clear solution to the problems posed by the oxidation of pyrite in uranium tailings comes from this study. Exclusion of air (oxygen) is a most important preventative; water cover substantially reduces the availability of oxygen. Preventing bacterial catalysis of the oxidation by using bactericides, by chemically breaking the chain of integrated oxidation reactions, by maintaining anaerobic conditions, or by maintaining a neutral or alkaline pH all substantially reduce the oxidation kinetics.

Removal of most of the pyrite by flotation will obviously reduce, but will not eliminate, the impacts from pyrite oxidation. Controlled oxidation of the remaining sulphide in the flotation tails, if this were possible, would provide an innocuous tailing so far as acidity generation is concerned.

No cost studies have been done on suggested options. These would be necessary before any definite choice between options is possible. Further research on bactericides, on the use of calcite (limestone), and on the effects of apatite (phosphate rock) additions is required. Knowledge of how the oxidation rate varies with the sulphide content is needed if flotation is to be further considered. Some tests are needed on the deep water disposal option; this approach has considerable merit so far as reducing the kinetics of pyrite oxidation.

The existing major deposits of pyritic uranium tailings in Canada, in Elliot Lake, differ from massive sulphide deposits treated by flotation, and from pyritic coal deposits. The relatively low sulphide content, the effects of leaching and the presence of radionuclides are three differences. However, close contact should be maintained with studies on other sulphidic tailings so the research being done on these tailings can be utilized where it is applicable.

This report is respectfully submitted.

Yours truly,

K. D. Hester

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EXECUTIVE SUMMARY

Special Boards were set up in the 1970's by the provinces of Saskatchewan and Ontario to examine the environmental impact of uranium mining in their respective jurisdictions. One recommendation by these Boards was that post-operative environmental concerns be more thoroughly addressed. The National Uranium Tailings Program was set up by the Federal government to examine these concerns. This report, funded by a National Uranium Tailings Program contract, examines the impact pyritic uranium tailings can have on the environment on mine abandonment, and considers practical approaches to reduce this impact. The impact comes primarily from the generation of large quantities of acid and acid salts from pyrite oxidation. The report considers pyrite per se, and not radionuclide emissions, except as they relate to pyrite oxidation. Because the Elliot Lake tailings deposits are the only large pyritic uranium tailings deposits in Canada, the report is developed around the Elliot Lake operations.

The conceptual approaches one can take with such a problem are considered. It is concluded that the most feasible option is assuming institutional care will be prolonged, and regulatory agencies will be able to define needed monitoring and control requirements. This is basically an extension of the assumptions intrinsically incorporated in the legislative framework under which the mines currently operate.

The criteria used to assess ways to treat pyritic uranium tailings to reduce the impact of acidic emissions are reasonableness, perceived cost effectiveness, and currently being used.

As background to the problem, the ore processing method is reviewed. Ore processing consists of a strong, hot, long residence time sulphuric acid leach on the ground ore to dissolve the uranium, removal and washing of the solids by filtering or thickening, and ion exchange recovery of the uranium from the resulting pregnant solution. The uranium is stripped from the ion exchange resin with an acid, and precipitated with an alkali. The dried precipitate is the "yellow cake" product. The active mines in the area also produce an acidic mine water which is treated for uranium recovery by neutralizing.

As defined in the final reports of the Environmental Boards mentioned earlier, current active operations have a well-defined treatment program for reducing the environmental impact of their effluents. The approach used is described. It is noted, however, that the long term effects of pyrite oxidation are not currently addressed in the program of the active mining operations.

The history of mining in the Elliot Lake area is considered. The ore bodies were discovered in 1953, and the first mine started in 1955. By 1958, twelve mines and eleven mills were operating in the area, producing some 30,000 tonnes of ore per day. But the government contracts which provided the funds to develop the mines were not renewed, and, after 1959, Elliot Lake faced a glum future. Many operations were shut down, and existing contracts were stretched out to maintain the town as an entity. For a time, government support was used to keep at least some operations going. Commercial viability seemed to be reviving in 1968 - 69, and the two remaining mining companies, Denison Mines and Rio Algom, began to expand.

During the interim period, from 1960 to 1970, the Ministry of Environment of Ontario had been monitoring the mining effluents. Under proding by the Ministry, treatment of acidic effluents coming from inactive tailings areas commenced in 1969. The current practice is described. The method consists of using slaked lime to neutralize the acidic run-offs, and barium chloride additions to precipitate radium. The resulting precipitates are settled in settling basins. This system prevents the impact of acidic effluents on the environment. The advantages and disadvantages of this approach are reviewed. The major advantage is that the impact of inactive tailings on local watercourses is significantly reduced. The major disadvantages are the considerable cost, the very long time the operation must be continued to maintain control, and the production of a voluminous, low density, slightly radioactive precipitate.

The residual tailings from an active uranium operation are examined in detail. The quantity of sulphide sulphur which will be present in tailings deposited on surface is calculated as it is now, and as it will be at the time of the assumed shutdown of Elliot Lake mining operations. Currently about 8% of the tonnage of potential acid producing tailings in Canada is in the Elliot Lake area. The potential production of acid from these Elliot Lake tailings would be about twice that which is required to leach the uranium. It is pointed out that only a small portion of this amount, probably only that amount of sulphide in the tailings above the water table, will oxidize at any significant rate. It is estimated that this will include the sulphide in the top two metres of tailings.

The research presently being done on pyritic uranium tailings is considered, as well as some research on pyritic tailings. The reactions which are believed to cause the production of acid and acid salts are considered in detail, and the mechanisms examined, particularly biological catalysis.

The acid generation potential of the inactive tailings which is now emerging from research on current inactive tailings sites is discussed. The Appendices to the report contain much information which has been supplied, primarily by Rio Algom, to the Ministry of Environment of Ontario and to the Atomic Energy Control Board. The work being done under Rio Algom's monitoring program allows the potential tailings impacts to be assessed, and will aid in developing remedial procedures.

Information from one inactive site, Nordic, is used to postulate what might happen when Denison and Quirke are shut down, if no post operational treatment program is applied. It is estimated that some 5,000 t/a of equivalent acid would be produced from the two tailings areas. This amount of acidic effluent would clearly have a significant negative impact on the Serpent River watershed. Even if more than 90% of the sulphide is removed by flotation, the environmental impact will still be substantial.

Treatments which have been suggested for pyritic uranium tailings are reviewed. Covering the tailings with man-made covers, with borrow material, and vegetative covers have all been suggested. Varying the vegetation in the area to accept the variation in acidity levels has been proposed.

Treating the tailings after leaching and solid-liquid separation is discussed. Methods here include modifying the physical laydown of the tailings to maximize run-off, and specific chemical additions to minimize the pyrite oxidation rate. Limestone additions will reduce the rate of oxidation because they maintain the pH at a high level. Bactericides such as sodium lauryl sulphate keep the oxidation rate low by controlling the bacteria which catalyze the oxidation at low pH levels. Phosphate ion, added by additions of apatite, disrupts the series of reactions which lead to pyrite oxidation by complexing with the ferric iron.

Some groups have suggested stabilizing the tailings by special chemical additions, such as lime, silicates or asphalt. These materials make an artifical "rock" which retains the material.

Removal of pyrite by flotation is discussed. The treatment of pyrite concentrate can vary. The concentrate can be used to produce acid, or it can be immobilized in some way which will prevent oxidation.

Underwater disposal of tailings, either deep lake cover or shallow cover, is another approach. The water cover substantially reduces the amount of oxygen available for pyrite oxidation.

Various process modifications may be possible. Preconcentration has been studied extensively; this would separate the ore into pyritic and non-pyritic fractions, but clean separations have never been achieved. Heap leaching is a possible processing method. Considerable pyrite is used up generating the acid and ferric iron for leaching, but much still remains after all the uranium is dissolved. Chlorine, and chloride based leaching systems have been studied; these have not been oriented to pyrite removal but to radionuclide removal. Some work has been done on nitrate systems; these seem to have advantages only in special cases, such as at Palabora, where separation of uranium from zirconium is an important process consideration.

Of the various possible approaches, several have been considered in more detail. Limestone additions to the tailings as they are produced, or after pyrite flotation may reduce the rate of pyrite oxidation substantially. The addition of adequate amounts of limestone would keep the pH level sufficiently high that biological oxidation would be prevented. The limestone would eventually be leached from the tailings by formation of soluble calcium bicarbonate from the CO_2 in the atmosphere. Addition rates for limestone would have to replace that leached.

Limestone could also be used as a replacement for the slaked lime now used in the present treatment system installed at inactive tailings sites. The difficulty with limestone is its low reactivity. However, modifications to existing systems could lead to its use.

Bactericides, such as sodium lauryl sulphate, and sorbates and benzoates, might be used to control pyrite oxidation rates in the short term. Slow release systems, using encapsulated bactericides, might control for several years. These methods are probably short term, systems which would allow other better control techniques to be installed.

Phosphate addition may be a very long term solution. Phosphates are inherently stable materials. They form very insoluble compounds with both radionuclides and with ferric iron. The level of apatite required may be substantial, but research in this area is needed.

Pyrite flotation obviously will substantially reduce the potential acid which could be generated by pyrite oxidation. Control of acid generation from the low pyrite tailings remaining would be simpler, but would still be needed. Only if some treatment technique could be developed which would remove most of the sulphide in the flotation tailings could this technique be considered very long term. Otherwise, oxidation control in the low pyrite tailings will be needed.

Underwater disposal seems the best technique of all for active mine wastes. A system is suggested in the text that should reduce the pollution from other chemicals, in a deep water disposal scenario. Keeping the level of ground water at or just below the surface of the tailings, and using muskeg above this to form a bog should minimize pyrite oxidation.

A synthetic membrane covered with available borrow material, which would not need any structural strength seems one option which could also reduce pyrite oxidation rates to near zero over the short term. The disadvantage is the lack of knowledge of the rate of deterioration of the liners available.

It is concluded that there are many methods for reducing the oxidation of pyrite in pyritic uranium tailings. The pyritic uranium tailings in Elliot Lake are unique in that the sulphide portion can be reduced an order of magnitude by flotation. Mines treating massive sulphide ore bodies by flotation do not have this option. Elliot Lake tailings are also unique in having no buffering material; it has all been removed by the hot, strong acid leach.

It is recommended that bactericides, phosphates and limestone all be studied further as aids to reducing the rate of pyrite oxidation in Elliot Lake tailings. It is also recommended that new process research be oriented to the sulphuric acid leach, but try to devise a way to oxidize the residual sulphide in low-pyrite tailings. This would enable flotation to be done before the leach, and separate treatment of the pyritic and low-pyritic fractions. If the sulphide in the pyrite tailings could be mostly oxidized, then flotation could produce an innocuous tailings, which may never require any treatment for acidity generation.

The Appendices contain a Statement of Work, an extensive bibliography, considerable data on existing inactive and active treatment sites in Elliot Lake, information from Ron Nicholson of the University of Waterloo on rates of oxidation of pyrite after limestone additions, and notes on discussions with experts in pyrite oxidation.

Acknowledgments are included for many of those who assisted in preparation of this report. Several with whom I held discussions are not mentioned by name; my thanks for their help.

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DRAWING: NUTP - H - 01 UNDERWATER DISPOSAL OPTION

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1.0 INTRODUCTION

A major uranium mining industry developed in Canada in the 1950's. In the 1960's, it was recognized in both Canada and the United States that uranium mine tailings posed some unusual environmental problems. In the 1970's, these problems were addressed by both the Cluff Lake Board of Inquiry in Saskatchewan and the Environmental Assessment Board at Elliot Lake in Ontario. Both these boards agreed that the treatments developed by the mining companies were adequate to protect the environment during the period of active operations. Both agreed that the effects of uranium tailings on the environment after mine shutdown were largely unknown, and that further work was needed to define the hazards, if any. The National Uranium Tailings Program was formed to respond to these post operational concerns.

Pyritic uranium tailings are a special case because they autogeneously generate acidic releases which can mobilize radionuclides as well as other heavy metals. Oxidation of the pyrite, both abiotically and under biotical catalysis, causes these acidic releases. Many studies have been made and are continuing on these tailings. No clearly defined practical shut down approaches have been formulated.

This report addresses the shutdown difficulties of pyritic uranium tailings and is funded by a National Uranium Tailings Program contract. The contract requirements are defined in a Statement of Work, included in Appendix A. Because almost all such tailings in Canada are in Elliot Lake, the study is directed towards the Elliot Lake operations.

The report discusses the possible options at abandonmnet from the point of view of institutional control. It reviews the current operating procedures in the Elliot Lake mines and mills, and the current tailings treatment systems at active operations. It considers why the Serpent River was significantly altered by the active operations in the late 50's and early 60's. It indicates the condition of the tailings at abandonment, and discusses the current lime effluent neutralization system being applied at inactive tailings sites. It then estimates the effect of abandonment, both with normal tailings and with the tailings surface reduced in pyrite by the application of pyrite flotation. Both estimates assume equilibrium conditions, and no applied treatment.

The report then briefly considers many of the possible approaches for treatment of tailings on abandonment, and provides a subjective critique of these approaches. It then chooses several different approaches, and studies them in some detail. Of these approaches, deep water disposal and flotation receive the most attention, as they appear to be the most developed systems. Several other possibilities are also considered; most are supportive approaches, not stand alone possibilities.

Conclusions are drawn from the study, and recommendations for further work are included. Five appendices are included which provide back-up for the information in the report.

It is not possible to clearly define a complete shut down scenario at this juncture. The report does indicate several possibilities that appear practical. None of the alternatives are inexpensive. All require some type of long term institutional control. No hard cost information is supplied for any of the options.

It is hoped this report will assist in developing an acceptable shut down plan for pyritic uranium tailings.

2.0 OPTIONS - SOLUTIONS

Before considering a "practical" solution to the problem of pyritic uranium tailings, it is useful to review possible close down options. These are summarized in Table 2.0 and are expanded on below. The "practical" solution should consider these options and also examine them in the broader societal context.

2.1 CLOSE DOWN OPTIONS

Do Nothing. Walk Away. This option has been the one adopted for pyritic tailings historically. Hawley (Ref 44) has tabulated data on abandoned tailings in Ontario which have been sulphidic, and which have generated and are generating acid and heavy metal salts.

Minimum Final Treatment. Then Walk Away. This option would envision some remedial action to slow oxidation and prevent dusting, such as adding limestone and fertilizers to the surface layers, revegetating, and possible contouring the ground to maximize run-off while minimizing erosion. After this, the tailings would be abandoned.

Minimum Final Treatment. Then "Perpetual Continuous Care". This option would be similar to the one now adopted for the current inactive tailings areas in Elliot Lake - a minimal treatment on mine shutdown followed by collecting and treating effluent to prevent stream contamination. The "perpetual care" aspect would include ground water monitoring, and diversion or pumping of contaminated aquifers or seepages to a treatment plant.

TABLE 2.0

POSSIBLE CLOSE DOWN OPTIONS

1. **Do Nothing. Walk Away**

   The historical precedent for sulphide tailings.

2. **Minimal Final Treatment. Walk Away**

   For Example: Surface contour the tailings surface, and vegetate.

   Then walk away.

3. **Minimum Final Treatment. Then Perpetual Continuous Care**

   For Example: Surface contour the tailings surface, and vegetate.

   Then collect and treat all acidic run-off from the tailings. Maintain the vegetative cover.

4. **Improved Final Treatment. Then Continuous Monitoring and Perpetual Sporadic Care.**

   For Example: Pyrite flotation with satisfactory disposal of pyrite concentrate, say underground, or by deep burial.

   Surface contouring, limestone additions, and vegetation of the low pyrite tailings.

   Then monitoring, with a prepared remedial package, ready for implementation under defined monitoring levels.

5. **No Pollution Ever.**

   For Example: Develop new process which cleanly separates sulphide as sulphur, and the radionuclides.

   Sell sulphur on the open market.

   Encapsulate radionuclides, and dispose of them as a high level waste, underground.

   Then walk away.

Improved Treatment. Then Monitoring and "Perpetual Sporadic Care" This option envisions working towards a permanent long term solution to pyrite oxidation by reducing it substantially in some manner which would be considered "long term". An example would be floating off the pyrite, and adding limestone to the low pyritic tailings. The pyrite would be burnt, or put underground, or buried deep beneath the tailings surface. Monitoring systems would be established and would operate intermittently but with reasonable frequency to determine if the system was getting "off track". In the event that monitoring determined that a problem was developing, or had developed, a planned remedial action could be immediately instigated before significant environmental damage occurred.

No Pollution Ever. This option would develop some system which will forever prevent the system from turning acidic. As well, the system would have to prevent any release of radionuclides, ever. It would appear that one such system would remove sulphides and radionuclides entirely, and store them in a "permanent" repository such as envisioned for high level radioactive materials. This could be considered an "ultimate" solution.

2.2 THE "PRACTICAL" SOLUTION

Pyritic uranium tailings are one of a group of materials which can give problems because they generate acidic seepages in concentrations which are inimical to aqueous biota. It is estimated that there are about 1.3×10^9 tonnes of sulphidic base metal tailings in Canada, covering some 9,000 hectares, which are current or potential acid mine drainage producers. In the U.S., it is estimated that more than 9,000 miles of streams and rivers are badly contaminated by acid mine drainage from old coal workings; 9×10^6 tonnes of acid are believed to be generated each year from the pyrite - containing material from U.S. coal mines. So in both Canada and the U.S., a lot of people are examining possible mitigating methods.

Any "Practical" solution must be one that lends itself to reasonable implementation, should be acceptable both to the regulatory agencies and to industry, and is relatively cost efficient. Without a detailed cost analysis and comparison of all possible alternatives, this latter aspect cannot be rigidly defined. Because such a program would be immense and very time consuming, it will probably never be done. Rather, individual choices will be costed and discussed. Thus there is bound to be considerable subjectivity in the choice of a "practical" solution.

Another facet of seeking a solution is land use. Presumably some restrictions could be placed on tailings deposited on surface for the very long term. These restrictions could include, for instance, prohibition of use of tailings solids for backfill around homes, or for railbed or roadbed ballast. Restrictions, however, should be specific to the actual tailings. The tailings should not be allowed to impact on the surrounding environment, and hinder development of alternative industries in the area, such as hunting and fishing and tourism. If land-based tailings deposition is accepted by the regulatory agencies, as it has been, it appears to imply that regulation over the long term is also acceptable. This indicates a long term acceptance of institutionalism.

As noted, much work is being done in both Canada and the U.S. on acid run-off from non-uranium sulphidic materials. It is becoming accepted that acidic emissions from these sources must either be treated at the mine or prevented. This seems to rule out either of the first two "walk away" options. Nor does it appear that the last option is a "reasonable" one, unless some very simple answer can be devised. So some option which requires "perpetual care" either continuously or sporadically, either a great deal or only a little, appears subjectively to be the choice, no matter how much we would like to strive for a completely care-free solution.

In summary, the following have been accepted as "practical" criteria:

1. reasonableness
2. perceived cost effectiveness
3. currently used
4. restrains or prevents emissions.

3.0 ORE PROCESSING

This section considers the current ore processing system used in Elliot Lake which produces a pyrite-containing acid leached uranium tailings. (See Figure 3.0).

3.1 MILLING

Consider an Elliot Lake uranium operation. The ore is crushed and ground, and the ground ore is settled and filtered on disc filters previous to leaching. Leaching is done at the highest possible density. At the completion of the leach, the acidity is adjusted by the addition of lime (Denison) or limestone (Quirke, Panel, and Stanleigh) to a pH about 2.0. The solids are separated from the liquid and the solids are washed. This is done in a six stage CCD line (Quirke), or in one stage of thickening, followed by two stages of rotary drum filters (Panel, Denison), or using thickening followed by horizontal belt filters (Stanleigh). The solution coming from the thickeners may be treated as is (Stanleigh), or be clarified (Quirke, Denison, and Panel), previous to ion exchange.

In the ion exchange circuit, Denison and Quirke use a nitrate elution system for recovering uranium from the ion exchange resin. The nitrate reports to the ion exchange barren effluent on the next loading step. This minimizes the possibility of barren recycle, as a high nitrate in the pregnant solution will cause uranium self elution during the cycle, leading to high losses of uranium in the barren effluent.

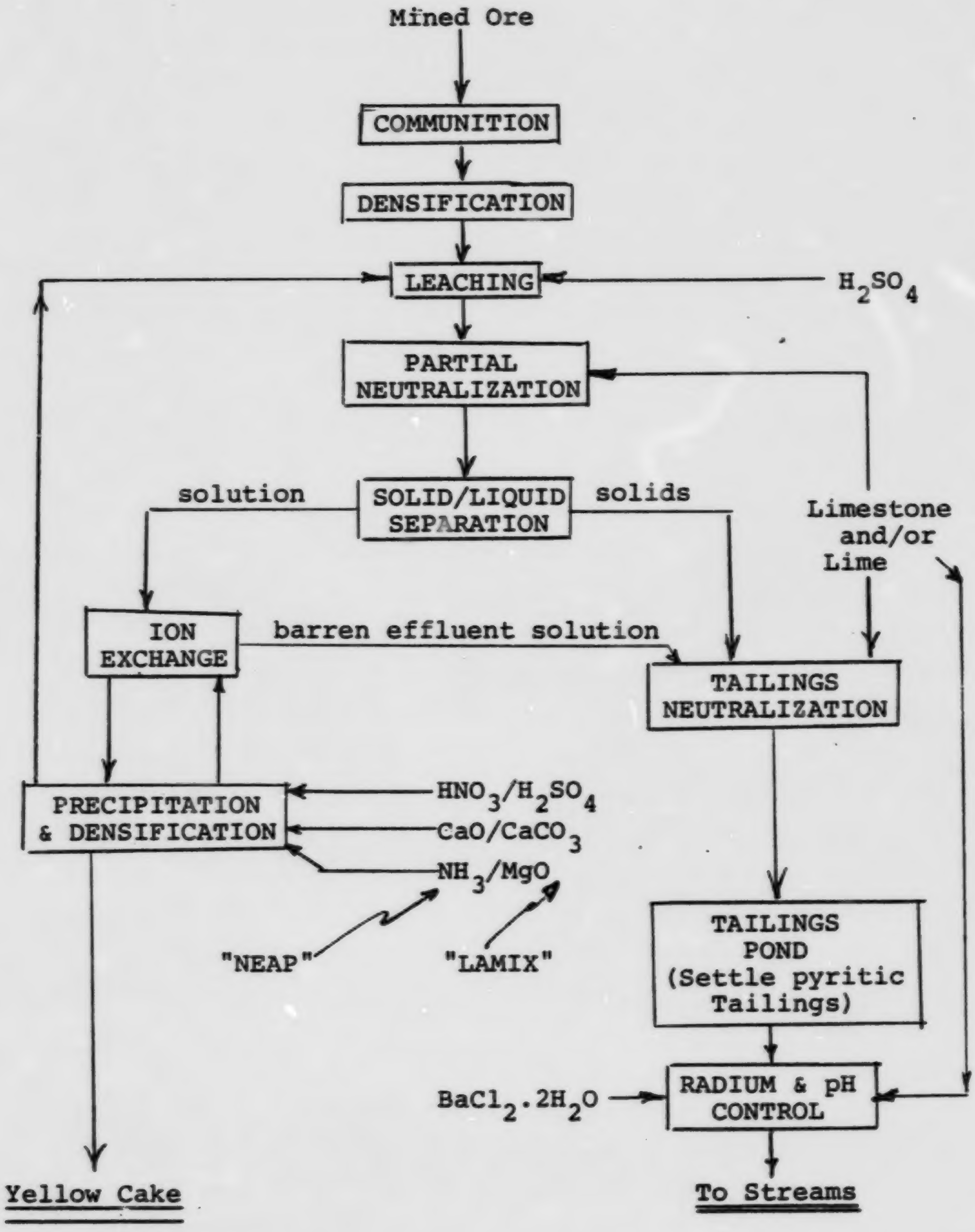

Mined Ore
COMMUNITION
DENSIFICATION
LEACHING
H2SO4
PARTIAL NEUTRALIZATION
SOLID/LIQUID SEPARATION
solution
solids
Limestone and/or Lime
ION EXCHANGE
barren effluent solution
TAILINGS NEUTRALIZATION
PRECIPITATION & DENSIFICATION
HNO3/H2SO4
CaO/CaCO3
NH3/MgO
"NEAP"
"LAMIX"
TAILINGS POND (Settle pyritic Tailings)
BaCl2.2H2O
RADIUM & pH CONTROL
Yellow Cake
To Streams

Figure 3.0
BLOCK FLOWSHEET - ELLIOT LAKE PLANTS

The Denison and Quirke "NEAP" system** results in ammonia and nitrate releases to streams as soluble chemical contaminants. The method uses a two stage precipitation system on the nitrate eluate. The first stage, called "iron removal", uses lime to control the sulphate in the eluant, and to knock out part of the acid. At the end of the lime addition, ammonia is used to adjust the pH to a sufficiently high value that thorium and iron in the final "yellow cake" meets refinery specifications. The so-called "iron cake" resulting from settling and filtering the mostly gypsum material produced is recycled to the leach circuit, because it contains a small smount of uranium. This returns nitrate to the leach, and thence to the pregnant solution, because of the nitrate liquor with the iron cake. The clear liquor overflowing the iron cake thickener is treated with ammonia to bring the pH up to about 7, which precipitates the contained uranium. This uranium-containing material is settled, and filtered, and finally dried, producing the plant product, an impure ammonium diuranante called "ammonia yellow cake".

Panel and Stanleigh use the "LAMIX" system*. Residual magnesium sulphate is the soluble chemical released to the streams. In this case, the uranium is eluted with 10% H_2SO_4. The sulphuric acid solution is partially neutralized, first to a pH of about 2.0 with ground limestone, and then to a pH of 3.5 with magnesia. The resulting gypsum contains some uranium, and is recycled to the leach circuit, after removal of the liquor by settling and filtering. The clear overflow from this gypsum removal stage is treated with magnesia slurry to increase the pH to about 7.0, and the resulting uranium cake is settled and filtered. The product is an impure mixture of uranium hydroxide and magnesium diuranate. It is called "magnesia yellow cake". Because in this (continued - 3.4 -)

** NEAP = Nitrate Elution -- Ammonia Precipitation

* LAMIX = Limestone - Acid - Magnesia - Ion - eXchange

system, most of the eluting ion is precipitated as insoluble gypsum, recycle is not practiced as at Denison and Quirke. Rather, the overflow from the gypsum thickener (iron cake thickener at Denison and Quirke) is returned to the mine water thickening circuit, with the overflow used for washing.

3.2 POSSIBLE TREATMENT VARIATIONS

This review of operations suggests possible alterations to the circuit which might reduce the difficulties arising from pyrite oxidation.

1. Flotation could be introduced on the feed to leach. A concentrate floated from the feed could be roasted to produce sulphuric acid, with the uranium-containing calcine being returned to the leach. Alternately, the concentrate could be pressure leached and filtered with the filtrate returning to the leach as lixiviant, and the cake treated separately from the main stream of ore.

2. Flotation could be used on the final washed solids from solid/liquid separation. The float concentrate could be roasted for acid manufacture, or pressure leached for acid making, then filtered with the lixiviant being recycled and the residue suitably treated. Alternatively, the concentrate could be discarded to some non-oxidative environment.

3. The final solid tails could be washed and treated, and pumped to deep water disposal.

4. Finely ground limestone and/or phosphate rock could be added to the tailings neutralization stage to restrict pyrite oxidation over time.

These suggestions are discussed in more detail in later sections of this report.

3.3 MINE WATER TREATMENT

In the underground mines at Elliot Lake, water is used to wet down the stopes and ore storage piles, and keep dust levels to a minimum. Water thus has a very real impact on the health of the miners by reducing levels of silica, and particulate matter in the atmosphere. The aeration of the water, and the presence of sulphur oxidizing bacteria has caused the water in all the mines to become acidic, and leach iron and uranium from the ore lying in the stopes.

At Quirke, Stanleigh, Panel and at the shutdown operation at Stanrock, which still treats mine water from underground, the procedure for recovering values from the mine water is similar. The mine water is pumped to the surface through acid-resistant piping. The water is neutralized by lime, and added to a high capacity thickener. Overflow from the thickener is used as wash liquor in the solid-liquid separation circuit in the operating mines. At Stanrock, this water joins the seepage from the tailings before it reaches the Moose Lake treatment plant, and substantially changes the acidity values of the liquid reporting to this plant.

The uranium-containing cake from the mine water treatment circuit is introduced to the leach in some manner. At Panel and Stanleigh, this cake is added to the leach discharge directly ahead of the partial neutralization circuit. At Quirke, the solids are leached in a separate circuit, and the solution added to the first stage CCD.

At Denison, the recovery of uranium is handled somewhat differently. The mine water is neutralized, and then added as make up water to the grinding circuit. This recovers the values, and minimizes the amount of make up water required for grinding. It also eliminates the necessity of thickening at Denison. Stanrock mine water thickener underflow is added to grinding also.

As well, at Denison, an underground leaching system is used to recover uranium from ore horizons which would be uneconomic to mine and hoist to the surface. Thus Denison is operating a system similar to that which was used at Agnew Lake. This keeps the environmental impact of the underground leached portion of the ore to a minimum by retaining the solid tailings underground, while still accruing the benefits from extraction of the values from the ore.

3.4 TAILINGS TREATMENT AT ACTIVE OPERATIONS

In all plants, the solids from the solid-liquid separation and solids washing step is combined with the discard barren effluent. This mixture is treated in pachuca tanks with lime to raise the pH to about 9.5. In some plants, the first part of the neutralization is done using ground limestone, which is more cost efficient, to a pH of about 3.5. Above this level, the slower reactivity of the limestone means the cheaper reagent is uneconomic. After neutralization, the mixed material is pumped to the tailings ponds for settling, and solids retention. In order to increase the line velocity in the tailings line, and prevent sanding out, some plants add water to the material going to the tailings.

In the tailings ponds, the solids settle, and the decant water runs to the treatment area. Here barium chloride, and sometimes slaked lime, is added to reduce the radium concentration and increase the pH (where this is necessary).

At Denison, the tailings are currently pumped from the mill around the periphery of the tailings pond to discharge at the end of the tailings basin remote from the decant. In this way, the excess alkalinity in the tailings slurry contacts the old tailings continually, and controls the generation of acidity. The decant from the tailings area goes through a decant well composed of a 48 inch reinforced concrete tailings decant pipe, and thence through the main dam. At the discharge of the pipe, a barium chloride solution is added to bring down the radium as a radium-barium sulphate precipitate in the 7 day residence time in the precipitate settling pond.

At Quirke, the tailings area has recently been expanded. Quirke has divided their tailings into several separate sections which they can treat separately as desired. These large "cells" allow each individual section of the tailings to act as a miniature tailings pond. Clear run-off from these smaller tailings ponds can be treated with barium chloride to precipitate radium within the tailings basin itself. The method is not beneficial for pyrite oxidation control. Quirke active tailings are currently turning acidic, and this necessitates lime additions at the tailings pond effluent outfall.

At Panel, the tailings discharge into what was the old Strike Lake tailings area. This had become acidic. The current discharge slurry, at a pH near neutral, discharges into one end of the tailings basin. The decant floods the old acidic tailings. The current run-off is acidic, at a pH of about 5.3 (1983 average - see Appendix C - III). Provision was made in the effluent discharge treatment plant to add lime to control pH as well as for barium chloride additions. The precipitate settles in lined settling ponds.

These ponds are baffled to maximize effective contact and settling time. Provision has been made for continuous pH monitoring, and the plant, which is of simple design, runs with no attendants.

At Stanleigh, the tailings are currently (April 1984) settled in the south west arm of the old Crotch Lake tailings basin. Run-off passes across the old tailings, and is still being treated with barium chloride at the temporary neutralizing plant built early in the 70's. The effluent from this plant combines with other drainages and receives final treatment in the modern sand filter plant. This plant, some 5 km from the Stanleigh mill, is remotely controlled by the mill operators. The pH is adjusted automatically with lime, barium chloride and flocculants are added, and the precipitate is removed on a mixed sand/coal filter bed. The backwash is applied automatically depending on the pressure drop. Backwash flushings return to deep water in the south east arm of the tailings basin. Eventually the old treatment plant will be phased out, and all treatment will be done in the new facility.

It can be seen from this review that pyrite oxidation remission or control is not specifically addressed during current tailings treatment. The deleterious effects of pyrite oxidation on streams in the area is, however, prevented.

4.0 DEVELOPMENT OF CONTROL PRACTICES

This section considers briefly the history of the Elliot Lake development, and current control practices at inactive mines.

4.1 HISTORY

The Elliot Lake ore bodies were discovered in the late spring of 1953. The initial Pronto outcrop which triggered the find was known to be radioactive but uranium values in the outcrop were low. It had been assumed the outcrop radioactivity was due only to thorium. However, when Frank Joubin obtained samples from a drill program, commercial uranium concentrations were found. We infer now that the surface uranium had been leached out by some naturally occurring leaching process acting over geologic time.

The geological perspicacity of Mr. Joubin led to the famous "back door staking bee" in the early summer of 1953 which opened up the enormous uranium reserves of Elliot Lake. Drilling programs rapidly got underway, and construction programs commenced in late 1953. The first mine, Pronto, came on stream in early September 1955 and the last of 11 mills commenced operations in 1958.

All the regulatory requirements of the day were met by the new operations. So far as tailings were concerned, each plant had its tailings neutralization circuit. Pronto commenced operation using limestone. It was soon realized this was an inadequate system, and in 1956 Pronto converted to quick lime, using the limestone ball mill as a slaker. All the other plants installed lime slaking equipment initially, and used slaked lime for both partial neutralization and tailings neutralization. The former was a process requirement. Anion exchange resin operated best on uranium adsorption at a pH of about 2.2, a free acidity less than 0.5 g/L. The solution leaving the leach runs about 100 times as strong - 50 g/L free acid; to operate the process, this had to be neutralized.

Only about half the acid added in the concentrators came out as "free acid". The remainder was tied up as "acid salts", mainly salts of iron and aluminium. Neutralizing these acid salts was not a process necessity. At the beginning of Elliot Lake it was seen by some operators as almost a waste of money. The minimum possible pH was used initially. Later, the pH reached in the tailings pachucas was increased to about 9.0 and currently a basic slurry is generally discharged to the tailings pond.

Pyrite oxidation was first recognized underground at Denison and somewhat later at Stanrock. Quirke mine water was noted as acidic in 1958. However, the acid generation potential of the tailings was not realized until after 1960, when several mines were shut down because of non-renewal of contracts. Previously, acid level increases in the streams were blamed on inadequate lime additions to the tailings neutralization tanks.

The Elliot Lake uranium mines went through a dramatic slump for several years after 1960. For a time, to keep the town of Elliot Lake operating, the Canadian government stockpiled uranium, paying the producers just enough to cover their costs. Simultaneously the Ontario Water Resources Commission (now the Ministry of Environment of Ontario) were documenting the analyses of mine effluents, and the degradation of the water in the lakes and streams below effluent addition points. (Ref 41). Given the state of the uranium market, it was obvious that tightening pollution control regulations would only lead to mine shutdowns.

In 1968 the uranium market began to recover. The Ministry of Environment of Ontario (MOE) and the Atomic Energy Control Board (AECB) began to tighten their control regulations, responding to public pressure. Plants to treat acid run-off from abandoned tailings areas were built at Nordic, (Nordic/Lacnor tailings), Pronto and at Crotch Lake, (Milliken/Stanleigh tailings) by Rio Algom in the early 1970's.

Additions of lime were made from a small plant at Panel to inactive Strike Lake tailings; the control approach here was perfunctory as it was recognized that this tailings area would soon be refurbished. A similar approach was taken initially at Stanrock (Stanrock/Canmet tailings) by Denison. Later in the 1970's a full scale plant to treat Stanrock run-off was built by Denison, after Denison decided not to mine directly at Stanrock. All these plants neutralized the acid run-off and added barium chloride solution for radium control.

As well, at the operating mines, provision for acidity control as well as radium control were provided. All four current operations - Denison, Quirke, Panel and Stanleigh - have provision to add both lime and barium chloride to their tailing pond effluents ahead of the precipitate removal system (settling ponds at Denison, Quirke and Panel, sand filters at Stanleigh). Only at Stanleigh has a provision for precipitate disposal been incorporated. Stanleigh recycles the gypsum and radium containing precipitate to the tailings area.

Thus the Elliot Lake operations have changed from using limestone alone as a neutralizing agent for tailings (Pronto) to using slaked lime. The minimum possible pH was used initially. Later, the pH reached in the tailings pachucas was increased to about 9.0 and currently a basic slurry is discharged to the tailings pond.

4.2 INITIAL ACIDIC ENVIRONMENTAL IMPACTS

The initial situation in Elliot Lake is best defined by a quotation taken from the report of a committee of Deputy Ministers in November of 1965. (Ref 55)

> "To ensure that suitable measures were taken at the outset to control the disposal wastes, the Department of Mines and Lands and Forests applied the provisions of The Mining Act and The Ontario Public Lands Act to the establishment of tailings (solid wastes) disposal areas, and, where applicable licences of occupation were issued for that purpose. The provisions of The Lakes and Rivers Improvement Act were applied and licences of occupation issued by the Department of Lands and Forests for the use of surface waters and for the construction of dams and control works for both water supply and waste disposal.
>
> "On the basis of the knowledge that was available when the mines were first developed, emphasis was placed on the control of tailings, employing practices that had been established and accepted as standard by the mining industry as a whole. The control of solid mining wastes by this operation was basically effective.
>
> "Chemical and physical examination of tailings impoundment discharges and of the receiving waters was begun in 1957 on a routine basis by the staff of the OWRC. Some minor losses of tailings occurred from time to time, but in most instances, corrective action was taken promptly by the industry. The pollution effects on receiving waters were minimized and confined to relatively small local areas adjacent to tailings disposal sites."

It should be noted that the OWRC was only formed in 1957.

While control of solid wastes was generally attained, the liquid effluents were the real problem. These liquids carried the radionuclides, which were the subject of the Deputy Ministers' report, down the streams. They also carried away the acidic emissions, which caused real damage to the stream biota.

A number of factors caused the significant environmental impact by the tailings effluents on the rivers and lakes of the Serpent River chain in the Elliot Lake area.

1. The lack of knowledge and experience about the environmental effects of uranium acid leaching was an important factor. Consulting engineers, government licensing bodies, mine management, and operators were unaware of milling impacts. For example, several mines took both potable and boiler feed water directly from lakes which received tailings effluents, not realizing there would be a rapid build up in total dissolved solids.

2. Add to this the rapid increase in tonnage milled. The first mill started in August, 1955. By mid-1958, about 10 Mt/a of ore was being treated in 11 mills.

3. The many lakes in the Serpent River watershed dampened the effects of pollution until it was far advanced and irreversible.

4. Some of the chemicals used, for instance, ammonia, are deleterious to fish at the level of concentration in the effluents.

5. As can be seen from the quotation from the Deputy Ministers' report, there was a general complacency about environmental problems. Mining had gone on for years in Ontario. The aberration of the Elliot Lake uranium mines was viewed as temporary, and quickly self-correcting.

6. Pyrite oxidation was not recognized as a problem. Pyritic waste rock was deposited at any convenient place, with no provision for seepage collection. At some operations, mine water was pumped directly to waste. Though current results (Appendix C-VII) show about 100 t/a equivalent acid per 1 Mt/a plant can be produced, the total acid release was probably low. Several mills shut down before acid generation underground started. Mild steel piping for mine water quickly deteriorated if the water wasn't neutralized underground. A several year incubation period is needed for the bacteria causing oxidation to be fully established.

7. All the plants had tailings neutralization circircuits, but the level of pH set at each mill varied considerably. Even conscientious operators maintained the lowest "reasonable" level, which was just over neutral. Most of the iron in solution was present as ferrous iron. This would not all precipitate at discharge levels of pH 7, or even slightly above. The oxidation to ferric iron in the tailings basins would generate acidity, and as there was no treatment of tailings pond effluents, the acidity would enter the streams. This was probably the major reason for the lowered pH in the receiving waters.

In February of 1959, the Ontario Water Resources Commission began monitoring tailings effluents (Ref 59). Weekly effluent samples were forwarded to the OWRC laboratory in Toronto, and the analyses were returned to the operators with comments. An average of the readings for all operating mines is given below.

Material	1959 Feb. 2 - Mar. 10	1960 Jan 20 - Mar. 14
Hardness (mg/L)	1,746	969
Total Fe (mg/L)	7.5	13.7
pH	5.5	5.8

Note that the iron level in 1960 (it would be ferrous iron) would reduce the pH to about 3.6 when it oxidized.

The uranium ore of the Elliot Lake area is mostly silica, with virtually no neutralizing minerals such as carbonates and phosphates. (Pronto is the one exception). Consequently, a stoichiometric relationship should exist between the acid added and the amount of neutralizing agent, because acid metal salts, such as iron and aluminium sulphates, still require neutralizing. Reagent additions at mills in the early days ranged as follows (Ref 56):

H_2SO_4	(93%)	67 to 100	lb/short ton
CaO	(92%)	33 to 46	"
HNO_3	(60%)	3 to 5	"
NaOH	(100%)	1 to 2	"

An average acid/base balance indicates some of the acid added was not being neutralized in the mill.

Levels of acidity are shown by the pH readings taken in 1966 from some of the streams running from the tailings areas. The data are from an OWRC report (Ref 56).

Denison	(operating)	4.4
Stanrock	(operating on mine water)	3.0
Milliken	(seepage into Elliot Lake)	5.1
Nordic	(operating)	4.3

Much further data corroborating these results are given in other OWRC reports (Ref 57, 43, 42, 41, 8)

In brief, a 3,000 short ton/day mill would treat about 950,000 tonnes/year of ore. About 40 kg/t of 93% H_2SO_4 would be added as the leaching reagent. The total of 100% acid would be about 35,000 t/y. About 9,000 t/y of this acid would go to solubilizing iron, mostly as ferrous iron. If the iron salts were not properly neutralized, some equivalent acidity would enter the Serpent River. In addition, acid might come from untreated mine water, and run-off from mine waste rock dumps. Both the acid in the mine water and in run-off from the waste rock would be from pyrite oxidation. But the initial drop in the pH in the Serpent River was not caused by pyrite oxidation alone, but also by inadequate tailings neutralization, and given the quantity of acid involved, this latter effect was probably the major one. It is not possible to define the impacts of early pyrite oxidation because the two effects are so intertwined.

4.3 CURRENT CONTROL STRATEGY - INACTIVE MINES

Current control strategies for acidic release **from** abandoned tailings areas only began at the end of the 60's. The Nordic operation typifies the system used. Nordic is a medium sized tailings pond containing some 12 Mt of tailings. The tailings are contained on two sides by rock hills, and on the other sides by waste rock embankments (Ref 18). Drainage from the area is collected in a ditch, which also picks up seepages through the tailings which come out the waste rock embankments. The flow of acidic liquor in the drainage ditch is conducted to a rigid flume with a metering weir. The exit section of the flume is baffled. Lime slurry is moved by mobile tanker from the lime slurry storage tanks in an operating mill. The slurry is transferred using the "acid egg" method to an agitated steel vessel inside a simple electrically heated housing. Barium chloride to precipitate radium can be added by bag to the lime slurry. Barium chloride additions at Nordic ceased on April 1, 1983. The lime alone seems to be maintaining radium control. (see Appendix C-V, and Table 5.3 - 1)

The lime slurry is circulated by a pump, with the pump discharging into a receiving box with several adjustable discharge nozzles. The excess material returns to the stock tank, and the bleed, at the rate determined by the nozzle size, falls by gravity into the front of the inclined flume. The baffles in the lower part of the flume thoroughly mix the lime slurry with the solution. The neutralized slurry discharges into a large settling basin. The solid precipitate of gypsum, iron and heavy metal hydroxides, and calcium (barium)-radium sulphate settles. The clear liquid passes through a rock cut and a swanp, and thence into the receiving waters.

This is a simple system to prevent emission of acid water, heavy metal salts and radium into the streams. One positive aspect is that control of these types of emissions is quickly obtained. This can be done rapidly and relatively simply, with a relatively inexpensive plant installation.

Another advantage which can be perceived is that the system allows the "natural" course of events to take place in the tailings at the same time controlling its effects. The top tailings layers are relatively rapidly cleared of sulphate, sulphur, and probably radium. This should eventually allow the surface to be covered with natural indigenous vegetation.

A third advantage is that the flow from the tailings (except for seepage) is readily monitored and easily controlled. What is going on within the tailings becomes apparent quickly, and the necessary remedial action can be taken.

One difficulty is the long term supply of lime slurry when all the mills are closed down. Quick lime cannot be used because of its relatively low reactivity. Bagged slaked lime can be used but is more expensive and requires a year round water source. The water source should be low in sulphate; otherwise some of the barium chloride mixed with the lime is precipitated as a sulphate and larger additions will probably be needed for efficient radium removal.

Another disadvantage of this system is the low density slurry produced. In the long term, settling ponds become filled with a gooey material which is slightly radioactive. One alternative would be densifying this slurry by recycling as has been demonstrated at Brunswick Mining. A slurry density of about 25% solids can be obtained. This slurry could then be "stabilized" and possibly stored underground at mined out uranium operations. Such additional processing adds to the operating cost but may be essential if the current system is prolonged.

A third disadvantage is that seepage from the tailings pile into adjacent aquifers or bedrock faults is acidic material with relatively elevated radium and iron analyses. The material in the ground water might move a considerable distance from the treatment plant before surfacing. The rocks in the Canadian Shield are relatively unreactive and differ greatly from those in the U.S. which have relatively high natural neutralizing capacity (Ref 15). So the tailings pile becomes a very long term risk to water quality in the general area, and requires continuous water monitoring studies.

In Elliot Lake this is probably more of a perceived problem than an actual one. Ground water flows through sand and gravel aquifers appear to be quite localized, and the bedrocks in the area are highly competent. An indication of their permeability comes from the underground operations. Mining is done beneath Quirke Lake and the mines are generally dry (Ref 9).

One other disadvantage is that it seems a considerable amount of the pyrite may eventually be mobilized. Downward percolation of liquid will move in air. The oxygen will penetrate the water table. All the pyrite at least down to the water table level will be attacked. According to Ivarson (Dr. K. Ivarson - personal communication) acid sulphate soils, found in several places in Canada, (Ref 11) generate acidity gradually but continually, and have been doing so for very many years. It seems possible that Elliot Lake tailings will do the same.

Current practice, then, has several positive and negative facets if it were to be considered as a long term solution. It is a "continuous perpetual care" option which would require some sort of institutional control. Disposal of the final precipitate should be more clearly defined. Because the system has been accepted by both the operators and the regulatory agencies, albeit on a short term basis, it must be considered a "practical" system for controlling the adverse effects of pyrite oxidation in uranium tailings.

5.0 ENVIRONMENTAL IMPACT - PYRITIC URANIUM TAILINGS

The environmental impact of the tailings is examined overall, with emphasis on the pyrite aspects in the following discussion.

5.1 INITIAL TAILINGS MATERIAL

The two different ion exchange processes at Elliot Lake will produce two different materials. Denison can be considered to epitomize the nitrate elution ammonia precipitation system. In 1982, (J.L. Chakravatti - personal communication) Denison's reagent consumptions show a stoichiometric excess of lime, which should provide some pyrite stabilization.

Nitrate and ammonia can be considered ephemeral pollutants. They do not form precipitates, so are quickly washed from the solids in the tailings basin. In a relatively short time, their tailing exodus effect should be over. Ammonia will still have down stream effects. (Ref 9).

The Panel operation can be considered to epitomize the LAMIX process. Panel reagent consumptions for 1983 (Bruce Arthur - personal communication) show almost a perfect stoichiometric balance over the year. Any excess acid generated would be neutralized by the lime addition step on the effluent from the tailings. Magnesium sulphate (Epsom Salts) is very soluble and should be reasonably quickly removed from the tailings. Because sulphuric acid is used for eluting as well as leaching, the LAMIX process adds a substantial extra loading of gypsum to the Panel tailings.

In broad terms, the reactions in the mill can be considered as neutralizing the approximately 40 kg/t of 100% acid used for leaching. In the partial neutalization stage, the excess acid, about 20 kg/t, is neutralized. In the tailings neutralization stage, the acid salts, mainly iron and aluminium, are neutralized. About half of the remaining 20 kg/t of acid, is tied up equally in these two metals. The net result is the production of about 55.5 kg/t of $CaSO_4$ as gypsum, (70.2 kg/t as $CaSO_4.2H_2O$) plus 7.5 kg/t of hydrates.

Assuming that fresh water is added to the tailings, and the tailings are pumped out at 30% solids, about 4.9 kg/t of $CaSO_4$ will be in solution as a saturated solution -- (2.1 g/L $CaSO_4$). If the solids settle to a consistency of 30% by volume tailings as void space (86% solids), then 0.6 kg/t of $CaSO_4$ will remain in the voids and 4.3 kg/t of gypsum will be discharged in the solution overflowing the basin. Precipitation, assuming a 50% evaporation rate, will probably increase the overflow solution volume by about 15% in the average tailings basin, adding another 0.7 kg/t of dissolved gypsum to the overflow. This will still leave the solid tailings containing about 50 kg/t of precipitated and dissolved gypsum.

If no pyrite oxidizes, then the gypsum will be gradually washed from the tailings. In general a saturated solution of $CaSO_4$ will run off. This seems to be shown by the lysimeter studies done in Ottawa (ref 28). The surface layers will be relatively quickly leached. Displacement of gypsum in lower layers will be slower. The displacement rate will depend on the porosity of the bottom of the basin, and on the compaction of the bottom layers of the tailings.

With no pyrite oxidizing, the radium question surfaces. The water passing across the tailings will now be moving through materials effectively void of sulphate. At this juncture there is no final resolution of the radium question. If "perpetual care" will be needed for the radium releases this could influence the "practical" approach to pyrite oxidation.

5.2 IMPACT OF PYRITE OXIDATION

The tailings basin at the moment of abandonment will contain about 5% precipitated calcium sulphate, and about 0.6% precipitated iron (as hydrate) and 0.2% aluminium (precipitated as hydrate). It will also contain about 3% sulphur as sulphide in pyrite. The potential impact of the tailing basins, at point of closedown, currently, and in the future are shown in the attached tabulation, (Table 5.2 - 1) assuming all the pyrite oxidizes to sulphuric acid. The overall acid generating potential is large. The level of 3% S can generate about 90 kg/t of sulphuric acid, more than twice the amount of acid added during treatment. Potentially, more than twice as much neutralizing agent will be needed to control the acid generation from contained pyrite as was needed during the original treatment of the ore, if all the sulphide is oxidized.

Dr. K.C. Ivarson, a soil microbiologist with the Canadian Department of Agriculture, has studied the iron-oxidizing bacteria extensively (Ref 11, 12, 13), and particularly their effect on forming jarosites and acid sulphate soils. Based on his comments and observations, there seems little chance that oxidation of pyrite above the water table in a tailings pond can be prevented.

Dr. Marvin Silver, working at CANMET has developed a lysimeter testing procedure to experimentally "speed up" the reaction which might occur in Elliot Lake tailings by a factor of about 9 (Ref 28). Core samples taken in one lysimeter box at the completion of the test period of 22.5 equivalent years show levels of about 3.5% iron below 7 cm compared to 2.5% above. The sulphur shows a similar pattern. This indicates there is still significant sulphide to create difficulties for many years to come.

TABLE 5.2 - 1

TAILINGS BASINS AND SULPHIDE CONTENTS
ELLIOT LAKE OPERATIONS

Basin	Mt tailings	Area Hectares	Estimated % S *	Estimated kt S
Pronto	2.1 - old U			
	2.3 - old Cu			
	4.4 - total	47	4.0	176
Lacnor	2.7 - old	24	2.5	68
Nordic	12.0 - old	107	3.0	360
Canmet Stanrock	5.5 - old	52	2.5	138
Spanish	0.4 - old	3	2.5	10
Quirke	3.7 - old	46	3.0	111
	28 - 1968-83	107	3.5	980
	13 - 1983-91	160	3.0	390
	44.7 - total	160		1481
Panel	3.3 - old	16	3.0	99
	5 - 1979-83	7	3.0	150
	9 - 1983-92	38	3.0	270
	17.3 - total	38		519
Milliken Stanleigh	7.5 - old	33	2.5	188
	65.0 - to 2020	116	2.5	1625
	72.5 - total	116		1813
Denison	? - old	2		
	39 -1957-83		4.35	1697
	77 -1983-03	250	3.0	2310
	21 -2003-10	?	3.0	630
	137 - total	?		4637

ALL VALUES APPROXIMATE.

* Sulphide Sulphur

Dr. J. Cherry has been associated for many years with field and laboratory research on Elliot Lake tailings, mostly at Nordic. One study (Ref 17) examined gas composition in a tailings section. Below 1 m, the O_2 level drops to the 2% range, with CO_2 reaching 1% (vs 21% and 0.003% in ambient air respectively). The O_2 deficiency shows vigorous oxidation of the pyrite occurs in the upper portion of the tailings. The high CO_2 is attributed to vegetation root emissions. This study also indicates the depth of the water table at Nordic varies between 1 m to 10 m, with a seasonal variance of about 1 m.

CANMET personnel in Elliot Lake have participated in similar studies. One (Ref 31) examines chemical profiles across the tailings. This shows how liquid migrates through the tailings, and how the concentration of contaminants builds up to "plume" from the base of the tailings pile.

Dr. M. Silver (personal communication) has studied thio bacillus populations in the tailings bed. High bacterial concentrations are found near the surface -- within the top few cm with bare tailings, down about 0.5 m with vegetated tailings. The population is sparce to just above the water table, where again high concentrations are found.

Dr. Robert Kleinmann and his staff at the U. S. Bureau of Mines Research Centre at Pittsburgh, Pennsylvania have spent many years studying the reactions occurring in pyritic coal mine wastes (Dr. Kleinmann -- personal communication). According to Dr. Kleinmann (Ref 20), the reactions proceed from the abiotic to those catalyzed by bacteria (Table 5.2 - 2). Control of the bacterially catalyzed reactions is the key to keeping the rate of pyrite oxidation under control. Dr. Kleinmann's research objectives have

TABLE 5.2 - 2

REACTIONS RESPONSIBLE FOR PYRITE OXIDATION

1. $2\ FeS_2 + 7\ O_2 + 2\ H_2O \longrightarrow 2\ FeSO_4 + 2\ H_2SO_4$
2. $2\ FeSO_4 + 5\ H_2O + \frac{1}{2}\ O_2 \longrightarrow 2\ Fe(OH)_3(s) + 2\ H_2SO_4$
3. $2\ FeSO_4 + \frac{1}{2}\ O_2 + H_2SO_4 \longrightarrow Fe_2(SO_4)_3 + H_2O$
4. $FeS_2 + 7\ Fe_2(SO_4)_3 + 8\ H_2O \longrightarrow 15\ FeSO_4 + 8\ H_2SO_4$

Stage I -- pH above 4.5, High $SO_4^=$, low Fe, little acidity.
Reaction 1. proceeds abiotically & by direct bacterial oxidation.
Reaction 2. Proceeds abiotically. Slows down as pH falls.

Stage II -- pH 2.5 to 4.5, high $SO_4^=$, acidity and total Fe increasing, low ferric to ferrous ratio.
Reaction 1. proceeds abiotically & by direct bacterial oxidation.
Reaction 2. proceeds at rate determined primarily by activity of T. ferrooxidans.

Stage III -- pH below 2.5, high $SO_4^=$, acidity, total Fe, and ferric to ferrous ratio.
Reaction 3. proceeds at a rate totally determined by activity of T. ferrooxidans.
Reaction 4. proceeds at a rate primarily determined by reaction 3.

From Reference 20

"Biogeochemistry of Acid Mine Drainage and a Method to Control Acid Formation" by R.L.P. Kleinmann, D.A. Crerar, & R.R. Pacelli. Mining Engineering - March 1981.

been to devise ways to reduce or eliminate bacterial action by direct focus on the living organism, thiobacillus, which catalyze the rate controlling reaction.

Dr. Alfred Stiller (personal communication) and his associates at the University of West Virginia in Morgantown, West Virginia have examined the problem from a different perspective. In their view, the critical reaction is:
$7\ Fe(SO_4)_3 + FeS_2 + H_2O \longrightarrow 15\ FeSO_4 + 8\ H_2SO_4$.
The bacteria produce ferric ion from any ferrous ion present, and this rate is rapid as soon as the hydrogen ion rises to the point where the ferric iron is soluble. (Table 5.2 - 2). The soluble ferric iron is the oxidant for pyrite. If the ferric ion could be immobilized, then pyrite oxidation would virtually cease. Dr. Stiller and his group have this immobilization of any generated ferric ion as their objective.

Dr. John Cherry and his research colleagues at the University if Waterloo in Kitchener, Ontario are examining means of preventing the bacterially catalyzed reactions from getting started (Ron Nicholson - personal communication). They have considered how the carbonate ion might slow down the rate of oxidation (Ref 26). Carbonate could be supplied by limestone additions. The limestone will be effectively mobilized by the formation of bicarbonate from carbon dioxide in the air (Ref 37). $CaCO_3 + H_2O + CO_2 \longrightarrow Ca(HCO_3)_2$.

In this regard, the increase of carbon dioxide in revegetated tailings would tend to increase the mobilization of added calcite.

Dr. Cherry's group have also examined the effect of porosity in pyrite oxidation (Ref 26) and attempted to quantify how porosity will effect oxygen permeation with concommitant pyrite oxidation. They have also considered how the oxygen partial pressure affects oxidation rates.

N. K. Dave and T.P. Lim at CANMET Mining Laboratory managed by Dr. Reini Tervo in Elliot Lake have considered that relatively complete exclusion of oxygen from the tailings will minimize or virtually stop pyrite oxidation (Ref 34). They have examined water cover as the method to exclude oxygen. It appears quite clear from their work and the work of others (Ref 9, 38) that pyrite oxidation virtually ceases when tailings are continuously covered with water.

Another approach would be the removal of pyrite from the tailings. The obvious first choice would be pyrite flotation (Ref 33, 27). The recovery of pyrite to concentrate could be high, 90% plus. However, the concentration of sulphur in the tailings would still be great enough to generate considerable acidity. Conventional pressure leaching would also leave a significant pyritic residue (Ref 23). Chemical conversion by using an oxidant in leaching does not appear to have been a research objective, although aqueous chlorine leaching will reduce the pyrite level somewhat (Ref 29, 30, 31).

Dr. Marvin Silver has suggested (personal communication) that reducing or eliminating oxygen ingress to the tailings may be the best approach. His work has shown that a layer of muskeg on the surface of the tailings reverses the trend to acidification of Elliot Lake tailings (Ref 28). If oxygen ingress can be prevented, then pyrite oxidation could be avoided.

In summary, many people have studied and are studying ways to prevent pyrite oxidation. The work is directed towards bacterial control, oxygen exclusion, elimination of the pyrite itself, and oxidation rate reduction.

5.3 ELLIOT LAKE SPECIFICS

It is not possible to separate the impact of environmental practices (Ref 8, 41, 42, 43) at active mines with the effect of inactive tailings drainage. As noted earlier, half the acid added in the plant goes out as acid salts, and originally the neutralization practices did not fully reflect stoichiometric requirements. As an example, most of the iron in solution in the leach is present as ferrous sulphate. If this is not precipitated, acid can be generated from the chemical oxidation and precipitation of ferric hydroxide. The ferrous hydroxide does not come down until pH of about 8 is reached, and many operations originally discharged tailings at substantially lower pH levels.

$$4\ FeSO_4 + 10\ H_2O + O_2 \longrightarrow 4\ Fe(OH)_3 + 4\ H_2SO_4$$

$$4\ Fe(OH)_2 + 2\ H_2O + O_2 \longrightarrow 4\ Fe(OH)_3.$$

In order to assess the impact of tailings on shut down, current operations at inactive tailings impoundments can be examined. Denison has one operation at Stanrock tailings area. Stanrock provides treatment of mine water by liming the pumped solution from underground and settling this in a high capacity thickener. Thickener overflow goes to the Moose Lake treatment plant. Feed to this plant does not, therefore, really reflect tailings run-off, as it includes solution which has already been neutralized. In examining the Moose Lake feed (J.L. Chakravatti - personal communication) it was obvious feed readings did not reflect Stanrock tailings effluent.

Rio Algom has treatment plants currently operating at inactive tailings areas at Pronto and Nordic (Nordic/ Lacnor tailings). A plant constructed about 1972 still operates at Crotch Lake (Milliken/ Stanleigh tailings).

Data has been tabulated in Appendix C on operation of the Pronto plant, the Nordic plant and from Panel and Quirke tailings areas. The help and co-operation of Rio Algom personnel for making this information available is gratefully acknowledged.

It should be noted that the volumes and assays provided in all the tabulations in Appendix C are not continually measured and assayed values. They express the results of intermittent measurements using relatively unsophisticated equipment plus occasional grab samples for analyses. They reflect considerable discussion with MOE, and clearly define what is happening. They are by far the most complete data available.

Appendix C - I gives the analyses for the fresh water at the head of the Serpent River system. Changes to these analyses will reflect the impact of current uranium operations and, on shut down, the effect of run-off from inactive tailings. It should be noted that the fresh water has low hardness and low buffering capacity.

Pronto is a relatively small low lying tailings basin with tailings from both the acid leach uranium operation and from a later copper operation which used flotation to remove a chalcopyrite concentrate. The run-off is restricted and the plant operates intermittently. The tailings area has been vegetated. The volume of run-off, the pH, the radium levels and the barium chloride additions from 1973 to 1983 are shown in Appendix C - II. The level of radium in the effluent has dropped substantially over the 11 year period. The pH of the feed to the treatment plant has remained relatively constant. The solution volumes vary substantially from year to year.

Readings over the whole area for 1976 and 1983 are also given. There does not appear to be a definite decrease in acidity levels, and the tailings have been weathered for nearly 20 years.

The average feed to the effluent treatment plant at Pronto is compared below :

	YEAR	
	1976	1983
Average pH	3.09	3.65
Average TDS (mg/L)	1005	622
Average $CaCO_3$ acidity (mg/L)	206	90
Average flow (L/S)	31	64

Results from the currently operating Panel plant are given in Appendix C - III. Panel operated from 1958 to 1961, and then recommenced operation in 1978. The plant treats about 3,000 t/d or 1.0 Mt/y. Substantial revisions were made to the old tailings basin. Tailings had originally been deposited in Strike Lake. When a broken tailings line spilled tailings into a valley south of Strike, it was allowed to discharge there continuously, filling the valley and spilling over into a creek at the far end. The new tailings system uses Strike Lake as the solids repository and provides a decant south of Strike which floods the old tailings. All these tailings had been generating acid previous to recommencement of the operation in 1978. Lime additions to the run-off were made, but the treatment plant did not treat the total flow.

The Panel data gives analyses for swamp water just north of the tailings pond; this is similar to the fresh water in Dunlop Lake. Panel uses Quirke Lake water for both potable and process requirements after a treatment including cation exchange and chlorination. The increase in dissolved solids between Dunlop Lake and Quirke Lake reflects the effluent additions from Quirke and Denison tailings.

The treatment plant feed at Panel is acidic (pH 5.3) probably reflecting the washing out of acid from the tailings. Panel does not over neutralize, so there is no protective alkalinity coming from the mill. The lime addition at the treatment plant removes acidity and provides a slightly alkaline effluent.

Appendix C -IV provides data from the currently active Quirke operation, presently milling about 5,000 t/d or 1.75 Mt/y. Data is provided for 1976 when the milling rate was about 4,000 t/d or 1.4 Mt/y, as well as for 1983.

Samples of Dunlop Lake water at two places are given; these water analyses can be compared in Appendix C - I. The Serpent River is sampled above and below the inlet of Quirke tailings effluents. The sample above Quirke effluent inlet reflects Denison tailings effluent addition to the stream. Denison uses Quirke Lake water as mill process water and for underground spraying. Quirke uses fresh Dunlop Lake water for processing.

The feed and discharge from the effluent treatment plant are provided. Quirke tailings were acidic in 1983 with a average of pH 4.2. This reflects the method of tailings disposal discussed earlier. The lime addition at the treatment plant brings up the pH but does not completely eliminate high iron levels. As noted above, this will lead to some acid generation down stream when the ferrous sulphate oxidizes.

Nordic data is given in Appendix C - V. Nordic has been a most intensely studied inactive tailings area.

Numerous papers (Ref 15, 16, 17, 26, 34) have been written on the various aspects of Nordic tailings.

Pertinent feed data for the Nordic plant is tabulated for the years 1975 to 1983 inclusive. As well, feed and discharge from the Nordic treatment plant is tabulated for the year 1976 and 1983. Information from Nordic is also shown in Table 5.3 - 1.

So far as acidity generation, Nordic is getting worse. The long term implications of this are reviewed in a later section.

The actual stream loading from the Quirke, Nordic and Pronto plants are compared for the month of May from 1976 to 1983 in Appendix C - VI. The data are not quite parallel but do indicate the possible impact of untreated tailings relative to an existing operation. As can be noted in the Nordic data, (Appendix C - V) May is the high water run-off month.

Mine water analyses at Denison, Panel and Quirke are compared in Appendix C - VII. The Quirke I data is from an inactive mine, which has been treated twice by flushing with barren effluent to remove uranium.

Appendix C - VIII provides a summary of operation of all the Rio Algom treatment plants for the years 1982 and 1983. This allows the flows, and the reagents consumed at active and inactive tailings effluent treatment plants to be compared.

In summary, pertinent data has been obtained, courtesy of Rio Algom, which provides a good assessment of the acidification which is occurring at inactive tailings areas, due to pyrite oxidation.

TABLE 5.3 - 1

LIME & BARIUM CHLORIDE ADDITIONS - NORDIC TREATMENT PLANT

YEAR	Vol Treated 1000 m^3	92% CaO Added tonnes	$BaCl_2.2H_2O$ Added kilogram
1974	?	384	6039
1975	1709	430	7087
1976	1109	421	6711
1977	1521	476	8156
1978	1140	655	12,869
1979	2444	1048	23,850
1980	1591	880	10,500
1981	1589	766	15,075
1982	2328	1032	24,550
1983	2123	1008	6500 *

* Barium chloride additions ceased March 31, 1983 because the feed radium levels had dropped to near 10 pCi/L & because lime additions provide a partial radium removal.

5.4 PREDICTED LONG TERM IMPACTS

Table 5.2 - 1 shows the estimated tonnage in tailings and the estimated sulphur content at shutdown of all Elliot Lake operations. The best estimate for total shutdown currently seems to be about 2020. At the end of 1983, about 112 million tonnes of tailings had been deposited in Elliot Lake area. There were 4 active sites totalling about 87 million tons and 5 inactive tailings areas totalling about 25 million tonnes. By 2020 tonnage of tailings will almost be 300 million tonnes. The area of the inactive sites is now about 233 hectares. The area of the currently active tailings areas which will become inactive by 2020 will be about 564 hectares, not including whatever area may be required for the last tonnages from Denison. If the underground leach program works very well at Denison, perhaps no further tailings area will be needed. In any event, the fresh tailings area which will be exposed is about 2.4 times the present inactive tailings surface and the total will be about 800 hectares.

This compares with about 9000 hectares containing 1.3 billion tonnes of sulphidic base metal tailings now laid down in Canada at various sulphide properties. Elliot Lake tailings currently comprise about 8% of the total sulphidic tailings in Canada.

The tonnes of sulphur currently in the inactive tailings is about 750 x 10^3 tonnes. The active mines on shutdown will have in the region of 8 million tonnes of sulphide sulphur, some 10 times the present amount. Any environmental impacts from pyrite oxidation will, however, be more likely to be comparable on an exposed area basis rather than a gross content basis.

The amount of acid which will be generated from these tailings areas is not known, but a reasonable estimate may be obtained by considering acid generation from existing inactive tailings areas. Nordic data are tabulated in Appendix C - V and Table 5.3 - 1. The Nordic tailings are constructed so that seepages can readily escape from one side of the tailings. These seepages are picked up and treated in the downstream treatment plant. The seepages migrate slowly, and probably the first three years of Table 5.3 - 1 give a reasonable indication of surface run-off. Lime additions averaged 412 tonnes/year. Including Lacnor, this would be run-off from about 131 hectares. Assuming all the acid was neutralized, and that the lime is 92% available CaO, this is an acid generation rate of 5 t H_2SO_4/ha/y.

In 1979, the lime additions at Nordic rose to 1000 t, and after dropping back for a couple of years, have been up to this level again for 1982 and 1983. This probably indicates that the seepages are now joining the run-off going to the treatment plant. The average water flow for 1982 and 1983 was 2,225,000 m^3/y, and average "$CaCO_3$ acidity" was 711 for an estimated acidity discharge of 1580 t/y of H_2SO_4. The average 92% lime addition for the two years was 1020 t/y, equivalent to 1642 t/y H_2SO_4. This is in good agreement with the assay estimate. The averaged acid generation rate was 12.3 t H_2SO_4/ha/y. This indicates a mobilization rate of about 4 t S/ha/y. These data are probally the best we have currently for prediction.

At shutdown, in a "walk away" situation, the water table in the tailings basin will assume some level, depending on run-off rates, seepage rates, the physical characteristics of the basin and its surroundings, and the rain and snow in the area (Ref 15). The location of this water table will be a specific characteristic of each specific tailings basin. It will also vary just as the water table does in soils. Using Nordic data, this water table ranges from a depth of 1 to 10 m, and varies by about 1 m (Ref 16). Bacteria have been found down to the water table, so it would appear that one could be reasonably assured that at least the top 2 m of any untreated tailings basin in Elliot Lake will generate acid. Assuming 30% voids in the top layer, and a 2.68 specific gravity for solids, each m^3 of tailings will contain about 1.9 t of solids, or about 56 kg of sulphide sulphur (at 3% S concentration). Each hectare contains 1120 t of S in 2 m depth.

At this juncture, the rates of percolation of oxygen into the soil is not defined; this flux will provide the rate controlling step for acid and iron salts generation at depth. Close to the surface, however, say the top 0.2 m, the rate will probably be more a function of biological activity than oxygen infusion. For many years the rate observed at Nordic would probably apply, and this can be used to estimate stream impacts.

It is easiest to examine these impacts using a hypothetical case, and the one considered is at the time of both Denison and Quirke shutdown. The current impacts are compared against the presumed impacts, assuming a "walk away" scenario - no treatment at each operation. This is obviously the best solution for all concerned, providing that Quirke Lake is not significantly downgraded.

Figure 5.4 - 1 shows the current impacts schematically. Precipitation is assumed at 80 cm/y (0.8 m/y) and evaporation at 40 cm/y. The area of the exposed tailings is expanded by 25% to form the assumed catchment basin,

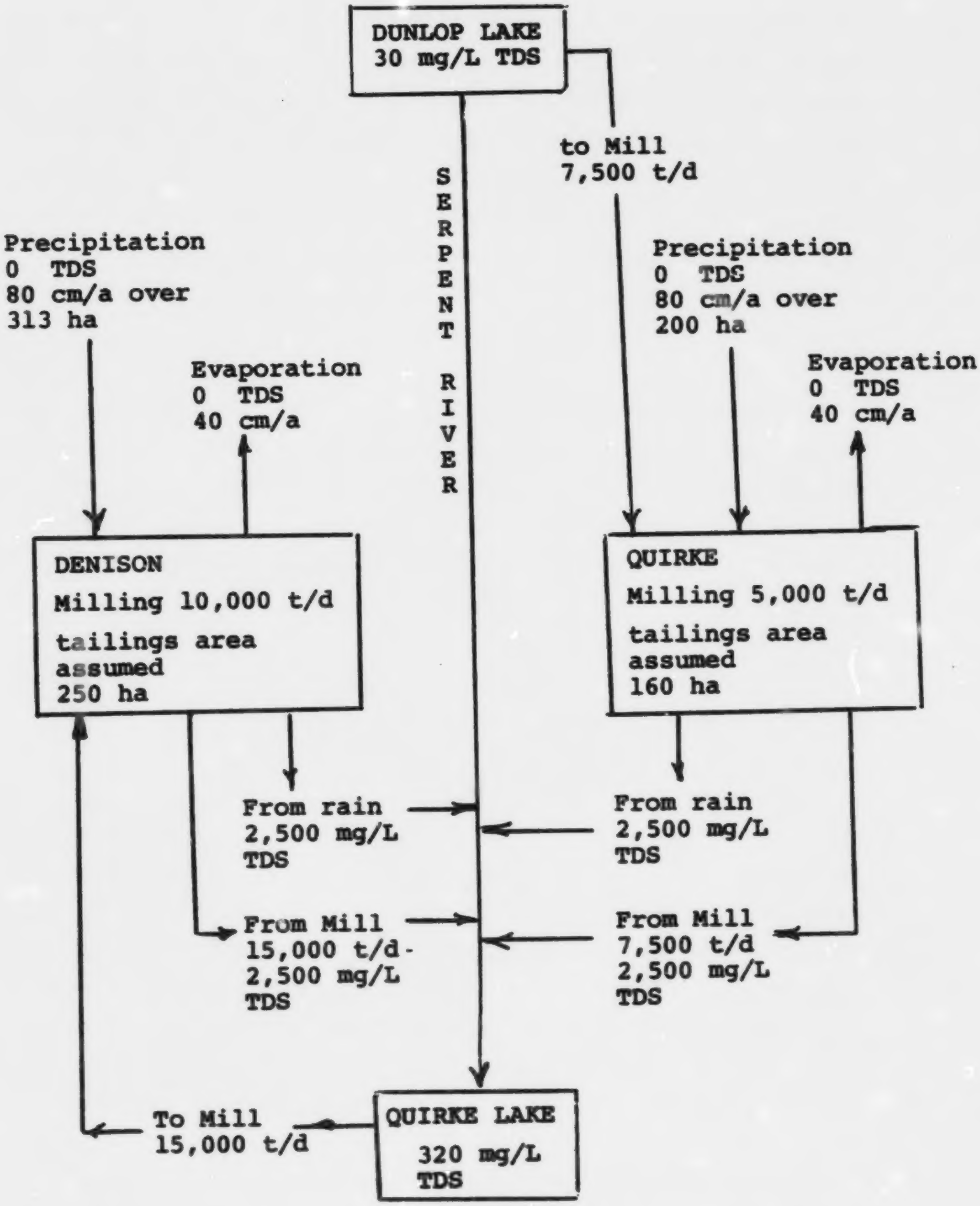

DUNLOP LAKE
30 mg/L TDS
to Mill
7,500 t/d
SERPENT RIVER
Precipitation
0 TDS
80 cm/a over
313 ha
Evaporation
0 TDS
40 cm/a
Precipitation
0 TDS
80 cm/a over
200 ha
Evaporation
0 TDS
40 cm/a
DENISON
Milling 10,000 t/d
tailings area
assumed
250 ha
QUIRKE
Milling 5,000 t/d
tailings area
assumed
160 ha
From rain
2,500 mg/L
TDS
From rain
2,500 mg/L
TDS
From Mill
15,000 t/d
2,500 mg/L
TDS
From Mill
7,500 t/d
2,500 mg/L
TDS
To Mill
15,000 t/d
QUIRKE LAKE
320 mg/L
TDS

Figure 5.4 - 1

PROTOTYPE TAILINGS BASINS

QUIRKE & DENISON

so effectively 1.0 m/y of precipitation will fall in this basin area, of which 50 cm/y will join the run-off. Downward percolation is assumed nil - if there is any percolation, it is assumed to get to the Serpent River by seepage. Quirke is assumed milling 5,000 t/d for 350 d/y and Denison 10,000 t.d for 350 d/y. In both cases it is assumed a 40% solids slurry is pumped to the tailings basin. No allowance is made for solution remaining in the tailings; relatively this will be small.

Denison obtains process water from Quirke Lake, and Quirke uses Dunlop Lake as its water source. The tonnages chosen above are similar to those actually being milled at the two properties. As a first assumption, one can consider any increase in total dissolved solids in the Serpent River as attributable to the plant effluents. The relative water flows can then be determined by the increase in TDS between Dunlop Lake and Quirke Lake, and running a material balance (see Fig. 5.4 - 1 and Table 5.4 - 1)

At shutdown, assume that the effluent from both Quirke and Dension is not treated, and no other remedial action is taken. The solid tailings should relatively soon generate acid at about the Nordic rate of 12.3 t/ha/y. The run-off will be from precipitation less evaporation. The estimated effluent from the 410 ha of the combined basins will carry about 5,000 tonnes of equivalent acid, or about 2.46 g/L of acidity. The iron level may be as high as 1.2 g/L. pH levels around 3. would be expected.

This solution will enter the Serpent River, and eventually get to Quirke Lake. The estimated effect on Quirke Lake is shown in Table 5.4 - 2. The acidity in Quirke Lake will increase to about 63 mg/L, substantially higher

TABLE 5.4 - 1

ASSUMED ACTIVE CONDITIONS
(Denison & Quirke)

1. **Denison Area**	
Precipitation	2,500,000 m^3/y
Run-off = evaporation	1,250,000 m^3/y
From Quirke Lake to tailings @ 15,000 m^3/d	5,250,000 m^3/y
Total Run-off	6,500,000 m^3/y
TDS out @ 2.5 g/L	16,250 t/y
TDS in @ 0.32 g/L (from Quirke L.)	1,680 t/y
TDS added	14,570 t/y
2. **Quirke Area**	
Precipitation	1,600,000 m^3/y
Run-off = evaporation	800,000 m^3/y
From Dunlop Lake to tailings @ 7,500 m^3/d	2,625,000 m^3/y
Total Run-off	3,425,000 m^3/y
TDS out @ 2.5 g/L	8,563 t /y
TDS in @ 0.03 g/L (from Dunlop L.)	79 t/y
TDS added	8,484 t/y
3. **Overall**	
TDS added	23,054 t/y
Increase in TDS in Quirke	0.290 g/L
Calculated average flow in Serpent to Quirke	80 x 10^6 m^3/y
Estimated drainage area above Quirke Lake	20,000 ha = 200 km^2

TABLE 5.4 - 2

ASSUMED SHUTDOWN & WALK AWAY CONDITIONS
(Denison & Quirke - No Treatments)

1. Assumed Effluent

(based on Nordic 1983 data)

pH	3.	
Acidity ($CaCO_3$)	2.46 g/L	
Fe	1.20 g/L	
TDS	3.5 g/L (?)	(may be higher)

2. Run-off Volume (both Denison & Quirke)

(precipitation less evaporation only)

2,050,000 m^3/y

3. Quirke Lake

Estimated Fe	31 mg/L **	
Estimated acidity	63 mg/L	
Estimated TDS	88 mg/L	(may be much higher)

** Note: If all the iron is ferrous and is converted to acid by the reaction:

$$4\ FeSO_4 + 10\ H_2O + O_2 \longrightarrow 4\ Fe(OH)_3 + 4\ H_2SO_4$$

then the pH should drop to 3.3 while iron will be below 1 mg/L.

Guidelines for mining effluents:

Fe -- less than 1 mg/L
pH -- 6.5 to 8.5
TDS - 250 mg/L

than at present. Even if Quirke Lake was dedicated to tailings, the outlet would still be far above the Canadian federal guidelines for mining effluents.

One treatment which has been widely suggested as a method of reducing the impact of pyrite oxidation is flotation of the pyrite from the tailings. No hard data is available on the comparative rates of acid generation from the unfloated tails, and the tailings after a pyrite float. Tests at Rio Algom on the tailings produced from a pyrite flotation test run at Denison (Ref 27) have shown that these flotation tailings will generate acidity.

If one assumes that all the water and air required are available at the surface of the tailings, then the amount of acid generated per unit time can be tentativley assumed directly proportional to the sulphide content. Flotation tailings will contain about 0.24% S, while the regular tailings will grade about 3.0% S. So the rate of acid generation from the flotation tailings should be only about 8% of that from the normal tailings. The acid generation rate would be about 1.0 t H_2SO_4/ha/y. The two tailings basins would generate in the order of 400 t/y of equivalent sulphuric acid. The effluent from the basin would run about 0.2 g/L of "$CaCO_3$ acidity" and about 0.1 g/L of Fe -- still a very acidic solution. Quirke Lake acidity would be increased to about 4 mg/L. If all the iron oxidized, the pH would probably be about 4.4. Again the outfall would not meet effluent guidelines. Some additional treatments are needed.

These are very simplistic calculations, assuming equilibrium conditions. It seems quite clear, however, that a "walk away" scenario would lead to the impact of the tailings spreading far beyond the tailings basin itself.

6.0 POSSIBLE TREATMENTS - PYRITIC URANIUM TAILINGS

This section provides a review of many of the treatments which have been suggested for pyritic uranium tailings. A subjective critique is included with these suggestions. Table 6.0 provides a summary of these approaches.

6.1 CONTINUOUS "ARTIFICAL" COVERS

"Artifical" covers have been considered for uranium tailings to prevent dusting and minimize the release of radionuclides. They would also minimize the entrance of oxygen and water to the tailings, and thus prevent the generation of acid.

Rigid cappings such as asphalt and concrete would require considerable preparatory work. In a tailings area of, say, 100 hectares, the project would be a major construction job. Tailings compaction, the preparation of the surface, the installation of a drainage system for disposal of run-off, reinforcing, and so on would all be required. Continual maintenance, just as on highways and shopping plazas would be needed to preserve the integrity of the capping. As noted in the discussions with Dr. Ivarson, bacterial activity could significantly damage the cover. Frost damage every year could also be expected.

TABLE 6.0

SUMMARY - PYRITIC URANIUM TAILINGS POSSIBLE TREATMENTS

1.0 Continuous "Artifical" cover. Cover the tailings with some man made material to exclude water & air.

2.0 Continuous "Natural" cover. Cover the tailings with a naturally occurring material to prevent air entering

3.0 Continuous "Vegetative" cover. Use vegetation in some manner to prevent pyrite oxidation.

4.0 "Vegetative" Area, Allow oxidation to proceed naturally, but use vegetation to dampen the impact of acidic run-off on streams.

5.0 Modified Process - Add-ons. Consider approaches which can be employed after the current milling process to prevent acid generation from the tailings.

6.0 New Milling Processes. Examine the effects of suggested processing changes on the oxidation of pyrite in the final tailings.

A synthetic membrane such as hypalon or flexible PVC could be installed, but would need a protective covering which would have to be non-sulphidic. Vegetation would "volunteer" on the covering, and, depending on its depth, the membrane might be damaged by tree roots or burrowing animals or insects. Frost movement, and settling of the tailings could eventually lead to membrane stretching, distortion, and failure. The conditions above the membrane, such as scouring from run-off, maintenance of the water table, drought effects on vegetation, would have to be assessed.

Nevertheless, synthetic membranes have been accepted by regulatory agencies as methods for building permanent "impervious" dams. A well designed and properly installed membrane, under a carefully laid thick soil blanket, would be an acceptable way of mitigating pyrite oxidation. Because such membranes are being used by the mining companies, they must be accepted as "practical" approaches. (Ref 53, 54).

A. Jongejan of CANMET (Ref 60) has suggested the use of briquetted compost from sewerage to cover tailings. The article cited considered only sewerage from Elliot Lake; this quantity would need augumentation. No work has been done on this approach regarding pyrite oxidation. However, if the eventual cover were several metres thick, presumably reducing conditions would exist beneath, effectively preventing pyrite oxidation. The system might be worthwhile exploring further, but is not considered a viable practice at this time.

6.2 CONTINUOUS "NATURAL" COVERS

"Natural" covers of rip rap, gravel, etc. have been used for reduction of radionuclide releases at depths of about 3 m at some sites in the U. S. In the case of pyrite oxidation, the effectiveness of such a cover would depend on its permeability. In the Elliot Lake area, clays are generally lacking. Clays would be important in making such a cover effective against oxygen ingress and relativley rapid solution movement. In the US, some studies on acid mine drainage examined bentonite additions to form synthetic clays. The objective was to compare bentonitic layers with PVC as a method of stopping pyrite oxidation. (Ref 54). The level of bentonite added (a maximum of 2 lb/ft^2 of tailings surface) was inadequate to form an impervious layer.

Because of the unavailability of suitable materials close to the tailings areas, the use of natural covers appears impractical in the Elliot Lake area. Where good cover material is available, the environmental usefulness of the method would require clear demonstration of its impermeability. The practibility would depend on the cost effectiveness of the method compared with other approaches.

If a "natural" cover was combined with a synthetic membrane, it's primary function would be to protect the membrane. In such a case, the loose soils around the tailings basins might be used as a cover. This use of a membrane plus a protective cover should provide an effective long term technical solution for pyrite oxidation in the tailings.

6.3 CONTINUOUS "VEGETATIVE" COVER

Vegetation of the tailings with legumes, grasses and trees has been demonstrated in Elliot Lake at several inactive tailings basins. In order to get vegetation to "take", the surface layer of the tailings must be neutralized and fertilized. Whether such vegetation will, in the long run, be self-perpetuating has not yet been clearly shown.

Vegetation clearly is desirable both to improve the appearance of the tailings areas and to reduce or eliminate dusting. Currently, this type of vegetation does not appear to reduce the acidification of the pyritic tailings (Ref 16). It is therefore an impractical approach to the pyrite problem, unless combined with perpetual continuous care, as discussed.

Dr. Marvin Silver, at CANMET, has studied the effect of a layer of muskeg over the top of a lysimeter operating on Elliot Lake uranium tailings. The test shows that a thin layer of muskeg is ineffective, but results from a 30 cm (about 8 inch) layer show that acidification can be prevented or stopped, at least in the short term and in a lysimeter test. Muskeg is a living material which covers many low lying bog areas in the Canadian Shield. It can grow in relatively acid - pH 4.5 - liquids. Because of its spongy nature, it apparently picks up and utilizes any oxygen in the water being introduced to the lysimeter, and prevents ingress of oxygen from the atmosphere into the tailings.

It's potential as a "practical" solution to the problem of pyrite oxidation in uranium tailings is examined in more detail in a subsequent section. (Ref 47, 53)

6.4 VEGETATIVE "AREA"

Ms Margaret Kalin of the University of Toronto Institute of Environmental Studies has suggested that a vegetative "area" might be established around a tailings basin. As one moved further from the tailings area, the vegetation would become less acid tolerant until finally it blends into the regular indigeneous vegetation.

This is an ingenious approach which tries to tie nature back into the control strategy. Verbatim comments from Ms Kalin are as follows: "In brief: Design the tailings such that natural recovery processes are facilitated.

"1. Abandonment Concepts

Depending on the tailings characteristics, the dormant site may in time generate environmental problems, such as acid generation, bioaccumulation of persistent substances, and others. A close out management program which considers long term biogeochemical changes, in the tailings itself and in the cover material (if present), may not only be effective in preventing environmental degradation, but also cost effective. Working with ecosystems in providing conditions which allow them to function is relatively easy, and the recovery occurs with least interference by man if given time.

"2. Close Out - Tailings Area

The effects of the close out (cessation of tailings discharge, preferrably concurrent with cessation of treatment systems) will result in changes in the tailings area, and in the zone which receives effluent waters. In the case of pyrite containing tailings, an akaline or neutral environment will change to an acidic environment wherever oxygen is present. It follows that the tailings should be deposited such that oxygen transport in the tailings mass is minimized.

"Oxygen transport may be reduced by promoting growth of biota during operation in the tailings pond which are tolerant of the conditions in the tailings, layers of organic matter may potentially reduce oxygen transport, active or passive. As tailings change from akaline to neutral to acidic environments, acid tolerant biota can be substituted. Organisms tolerant to acidic conditions would dominate the tailings in the long term.

"3. Close Out - Downstream Requirements

Locate the tailings area such that the effluents are received in basins large enough to facilitate the development of acid tolerant ecosystems. Downstream, these ecosystems would gradually decrease in acid tolerance, e.g. from a muskeg, to a cattail wetland, to a more mixed wetland and so on. The ultimate discharge could resemble relatively natural conditions, and the tailings deposition in the long term may only have resulted in diversification of the ecosystem types in the area.

" IN SUMMARY: Technological and engineered environments can be difficult to control in the long term. The multitude of environmental tolerances, adaptations, repair mechanisms and recovery processes which have evolved naturally can assist in tailings design. Abandonment concepts should be developed to work with nature and not against her. "

The difficulty with this concept, on the tailings basins as designed, are several:

-- All the "large" tailings areas in Elliot Lake will be elevated, and a damp or watery cover would not be possible to maintain naturally.

-- The basins will be designed with one effluent discharge area. This will contain strongly acid material. It would take a very large "swamp" system to accomodate the effluent - a system which would have to be constructed at considerable cost.

-- The concept would be a method of dispersing radioactivity and pyrite pollution rather than containing it.

-- Releases of acidity may cause the pH to drop to 2.5 or even lower. Even muskegs and bullrushes are not tolerant of acid levels this high.

By itself, this approach is not considered practical, but the idea might have merit if incorporated with other options. (Ref 47)

6.5 MODIFIED PROCESS - ADD-ONS

A number of "add-ons" to the existing process have been suggested. These can be divided into five different approaches: physical methods, chemical additions, tailings stabilization, pyrite flotation, and underwater disposal.

6.5.1 Physical Add-ons

The physical systems include tailings contouring, densification, and compaction.

Dr. Eli Robinsky of E. J. Robinsky Associates has suggested the "thickened tailings" system of disposal. In this approach, the tailings from the mill are settled in a thickener. The thickened tailings are then dispersed from a high point in the tailings area, and form a "cone", at their natural angle of repose. The advantages, according to Dr. Robinsky are: (i) the tailings are laid down relatively homogeneously, that is, fines and coarse come down together, (ii) large peripheral dams are not required, and (iii) the drainage from the laid-down tailings is generally more rapid, and off the tailings area, reducing seepage into the body of the tailings. (Ref 3)

The system has been applied at Kidd Creek Mines in Timmins. (C. Chapman - personal communication). Because of the muskeg, soil, and varved clays in the area, well built dams would have been inordinately costly to construct. By building a ramp to the central discharge area, and spigotting from there, Kidd Creek has disposed of considerable tailings mass with relatively minor dam construction. The method of disposal has led to considerable acid generation, however. Kidd Creek tailings run about 25% pyrite, and the run-off collected behind the low peripheral dams is very acidic (pH's as low as 2.0).

It is obvious that the "thickened tailings" system does not prevent acidification from occurring.

As the "thickened tailings" discharge is raised, increased density in the feed stream to the spigotting system minimizes the required area for tailings disposal as it steepens the angle of repose of the tailings, resulting in more rapid run-off, and lowered seepage. One equipment manufacturing firm (J. Elliott - personal communication) is working on a relatively simple and unique system which might be used to improve densification close to the spigotting area. This might permit pumping a manageable 50% to 55% solids slurry, and applying a 70% solids slurry to the spigots, recycling the water as needed. Overall, the surface area for a given mass of tailings is reduced, which should lessen pyrite oxidation potential.

Physical placement of tailings might be combined with compaction. The objective would be to minimize pore volume, and thus reduce the rate of percolation of water and air through the tailings mass. University of Waterloo personnel are examining this approach. (Ref 26) Minimizing pore volume should reduce rates of acidification, but there is doubt if compacted pore volumes could be maintained once acidification began, based on the expansion which occurs when bacteria become operative. Also, frost effects could change the porosity of the tailings over time.

It thus appears that none of the physical systems, taken alone, are practical methods for reducing tailings acidification. They would be useful, however, in conjunction with other approaches.

6.5.2 Chemical Additions

The current method for handling the inactive tailings sites is to collect all seepage and run-off and send this to a treatment plant. Here slaked lime is added to neutralize the acid and acid salts, and barium chloride is added to precipitate the radium. Ground limestone should be considered as an alternative to lime in this application. The reaction rates are considerably slower, but the reagent costs should be much less.

Limestone could be mixed in with the tailings to slow oxidation. If adequate dispersed basic material is present in the tailings, rapid biological oxidation of the pyrite should not occur. A stoichiometric quantity of limestone (about 10% by weight) should provide a protective buffer for many years. Because carbon dioxide in the air forms soluble bicarbonate, in the long term the protective calcite will be removed. The protective buffer could be replaced ("perpetual sporadic care"), or possibly the acid generation rate might be slow enough to be acceptable.

The advantages this approach has over letting the tailings acidify and treating the run-off are (i) no acidic seepages which could contaminate aquifers below the tailings, (ii) ground limestone could be used as the buffering material, inherently much less expensive than lime, (iii) much less "perpetual maintenance" input. Current studies may allow prediction of the time frame, and the required quantity of limestone needed (Ref 26).

Limestone additions to the tailings, and as a substitute for slaked lime appear to have practicality, and are examined in more detail in the next section.

Phosphate will form very stable complexes with some of the radionuclides, and with ferric iron. Phosphate complexing has been suggested as a method for "rock forming" - preventing the release of radionuclides, particularly radium - by Dr. W. S. Fyfe at the University of Western Ontario (Ref 35).

Dr. Alfred Stiller and his associates at the University of West Virginia have been using the addition of phosphate rock (apatite) as a way to stop the generation of acid and iron salts from pyrite. (Dr. Stiller - personal communication). The method has been used on pyritic waste materials from coal mining operations. Some West Virginia coal seams are topped with a pyritic layer. This layer is stripped off, the coal removed, and the pyritic waste returned to the barren pit. The waste is treated with finely ground phosphate rock for pyrite oxidation control (Ref 48).

Since phosphates may tie up both radionuclides and the iron salts resulting from pyrite oxidation, phosphate rock additions may have an application in Elliot Lake. This is examined in more detail in a later section.

Testwork at the Pittsburgh Research Station of the US Bureau of Mines has led to the development of anionic detergents as controls for bacterial oxidation. (Dr. Robert Kleinmann - personal communication). Again, coal mine wastes are the objective. Treatment of coal waste piles with sprayed on solutions of detergent have substantially reduced the rate of acidic production by acting as bactericides. The detergent used is presently sodium lauryl sulphate (SLS), although other detergents have been tested.

A particular advantage of SLS is that it is biologically degradable. During the current application tests, it has not appeared in significant amounts in tailings run-offs. (Ref 18, 19, 20)

While constant application of SLS to Elliot Lake tailings would not be a practical control measure, in certain circumstances SLS may prove to have an application. Its use is reviewed in more detail in a subsequent section.

The Pittsburgh Research Center has also done some experimentation with the salts of organic acids as bactericides. Benzoates and sorbates can be included in the pyritic wastes, and release slowly, and in a manner which restrains the biological oxidation of pyrite. This research is current, and whether these chemicals will prove more effective than SLS in the long term is not known yet (Ref 52).

6.5.3 Tailings Stabilization

Considerable interest has been expressed in the use of chemical stabilizers for the tailings. It was believed these systems would prevent mobilization of both heavy metals and radionuclides, and also prevent pyrite oxidation.

The Stablex Corporation in the U.S. provides the "Sealosafe" process which "relies on a crystal capture mechanism, in which pollutants present in solution form chemical bonds with the reactants creating a polymerlike matrix in which pollutants are trapped". In discussions with Stablex, (R. Podsiadlo - personal communication), the cost of the raw materials seem to be near $10/t of tailings. The system appears to incorporate the pollutants into a low grade cement. Several plants are operating on industrial wastes, but none on tailings.

Canadian Waste Technology was a company developed around the patents of David Krofchak. The company has met financial problems, and is currently (April, 1984) in the hands of Clarkson and Company as receivers. The technology developed by the company may be applicable to the stabilization of pyrite and radionuclides in the tailings (E. Seraphim - personal communiction). The approach is based on using the silica in the ore as a raw material, and tying up the tailings as an artifically formed rigid silicate. Some tests have been done by Denison on the method, and at one point the MOE was pushing the idea of using the approach. The reagent costs are said to be low, but royalty arrangements might increase costs substantially. Because of the secrecy surrounding all the company dealings, at this juncture stabilization using the Krofchak method is titillating, but cannot be considered a practical alternative.

Studies by the Gulf Research & Development Laboratories in Pittsburgh developed a system tailored to the "below ground" trench disposal method which Gulf would require if it developed a New Mexico mill. (Ref 64). The additives which were tried were fly ash and kiln dust. Addition rates were 3%, 6%, and 10% by weight of solids at varying solid weight % levels from 40% to 55%. Gulf personnel concluded that the density of solids was a major factor in reagent consumption, and concluded that fly ash additions at a level of about 6% for a 50% solids slurry were optimum.

Researchers at ORF in conjunction with CANMET personnel reported on several stabilization methods at the 1982 CUPMC meeting in Bancroft. (Ref 62). Addition of reagents ranged from 5% by weight of tailings to 50% by weight. Costs range from $2.00 per ton to $420.00 per ton. None of the systems could be considered as completely effective. The systems included the Krofchak system, which was the least expensive, and the lowest in addition rates to the AECL system of bitumenization, which was the highest cost, but also the most effective. The systems examined included two lime based systems, a silicate based system, and a bitumen based system. None of the approaches examined the effectiveness of the stabilization methods on pyrite specifically.

It can be generally concluded that the stabilization systems developed up to now are not of much use when one considers the quantity of tailings which must be addressed, and the ineffectiveness to date of the method.

6.5.4 Pyrite Flotation

The flotation of pyrite from the tailings has been advocated by many as a simple way to lessen the pyrite oxidation problem. Thus the Environmental Protection Service of Environment Canada advocated flotation in a submission to the Environmental Assessment Board during the EAB hearings in Elliot Lake. (Ref 65)

So far as non-polluting disposal of pyrite concentrate is concerned, underground disposal will probably soon be the simplest system. As the bottom areas of the mine become exhausted, pyrite concentrate could be used to fill old abandoned stopes. Another system, burial in the tailings themselves, may also be an option.

The cost of a flotation plant is considerable, both the capital cost and the operating costs. If some use could be made of the concentrate, there may be an incentive to install flotation. Thus the companies have recently completed a joint study on roasting and acid production. To be effective environmentally, acid production would have to be about twice utilization - and this in a market with an over supply of acid. It was decided acid production was not economically viable. (Ref 66)

An alternative, which combines disposal and acid production, is pressure leaching the concentrate to produce acid, and filtering the acid off and disposing of the remaining concentrate. This method has never been fully explored, but it might have some merit. It should be less expensive than an acid plant, and it could replace the leach acid requirement. (Ref 23)

It has been clearly demonstrated that flotation of pyrite from the tailings does not stop the generation of acid, but it should substantially reduce the rate of acid generation, and it certainly reduces the potential for acid generation. At present there does not seem to be data which allows the rate differences to be quantified.

Because of the substantial reduction in acid producing potential, a flotation plant should include some approach to immobilize the remaining pyrite in the flotation tailings. Restricting pyrite oxidation by calcite and/or apatite additions may be one approach. Another method would be the addition of a vigorous oxidant to convert all the remaining sulphide to sulphate. If this latter could be done, acidity releases from such tailings would never be a problem.

Flotation has been demonstrated to be relatively effective in reducing the pyrite available for oxidation. It could be applied to an operating plant with assurance. Costs are well defined. The method, combined with the addition of neutralizing chemicals, may be a "practical" way to prevent the tailings going acid. It is discussed in more detail later.

6.5.5 Underwater Disposal

There are two aspects of underwater disposal: deep lake disposal and water cover.

Deep lake disposal of tailings into Quirke Lake was first suggested by the mining companies in Elliot Lake some years ago. Currently study of the approach is at an impasse. (C. Chakravatti - personal communication - March 1984.).

Deep water disposal in Quirke Lake might have been a good approach for all the tailings at the beginning of the Elliot Lake camp. Several years back, the idea was vigorously opposed by the regulatory agencies. They believed surface disposal would allow the monitoring of pollutants, and more rapid response if problems developed.

Now that surface tailings basins have been assigned, and the construction techniques and disposal systems agreed to by the regulatory agencies, there are only two good reasons for re-examination of deep water disposal: (i) if there is inadequate capacity in the designated basins or (ii) if there is a substantial financial/environmental incentive for going to deep water disposal.

Studies have centered on pumping all the neutralized tailings to Quirke Lake, and allowing them to settle. The infusion of high dissolved levels of radium, plus the high levels of dissolved salts suggest that: (i) all of Quirke Lake outfall be diverted to a treatment plant, and (ii) to minimize the size of this plant, as much fresh water inflow as possible be diverted. Both these requirements would lead to very large capital and substantial operating costs. Both would have considerable negative environmental impacts.

For these reasons, it would appear that pumping all the tailings into Quirke Lake directly is impractical (Ref 9). So is reslurrying tailings in existing surface basins, and transporting them to Quirke Lake.

The advantage of deep lake disposal so far as pyrite is concerned is that the low oxygen content in the ambient water, plus the relatively low diffusion rate of oxygenated water into and out from the tailings mass should lead to negligible generation of acidity from pyrite oxidation. The migration of radionuclides from the mass of underwater tailings into the lake water should also be extremely slow, even if pore water concentration of nuclides is high (Ref 53). The perceived disadvantage of deep lake disposal is that if remedial action for pollution were required, it would be more difficult to accomplish.

A possible option for deep lake disposal which may have practicality is treating only the cake from the last stage of the solid/liquid separation and solid washing circuit. A small amount of these solid tailings would be combined with ion exchange barren effluent, and go to an existing surface tailings basin. The bulk of the solids would go to deep lake disposal, but without the liquid contaminants.

This approach uses the same treatments as now, so should provide no worse outfall from Quirke Lake than at present. It does allow the operation of the surface disposal areas to be substantially prolonged, so a larger surface basin is not needed even with an increase in tonnage milled. The concept is discussed at more length in a later section, as it appears to be one possible "practical" solution which may very well have substantial economic benefit (Ref 10).

Water coverage per se seems to have a substantial beneficial effect, even if deep lake disposal becomes a non-option. If water cover can be kept at or close to the surface, muskeg cover might be induced to shut off the oxygen supply, and prevent the tailings pyrite from oxidizing.

The saturation level of oxygen in water at 20 C is about 0.009 g/L. A litre of air at 20 C contains about 0.26 g of oxygen. Assuming the formula:

$$FeS_2 + 7/2\ O_2 + H_2O \rightarrow FeSO_4 + H_2SO_4$$

it would take over 100 L of oxygenated water to oxidize 1 g of pyrite, but only 3.6 L of air containing 20% O_2, assuming 100% oxygen utilization. The much lower availability of oxygen from water than from air means that pyrite oxidation below the water table is much slower than above it. Because water is needed for oxidation to take place, variations in the water table will lead to pyrite oxidizing in the wetted zone. But control of the water table by feeding water to the tailings basins during the dry season will reduce the mass of tailings which will become susceptible to rapid pyrite oxidation. This control could be readily used, and would augument provision of a non-sulphidic cover on the top of the basins at shutdown. In any final shutdown plan, the judicious use of water as a method for damping the rate of pyrite oxidation should be considered. (Ref 53)

6.6 NEW MILLING PROCESSES

A number of new processing approaches have been suggested for treating Elliot Lake pyritic uranium ores. Most of these developments are from Canadian sources.

6.6.1 Depyritization Options

Pressure leaching of the Elliot Lake ores was one of the original options explored for processing. Pressure leaching autogeneously generated the required acid for leaching from the pyrite contained in the ore. The work was primarily done by Dr. Frank Forward and his group at UBC, in conjunction with Sherritt Gordon Mines. Pilot plant studies were completed in Ottawa. The system worked satisfactorily, but only about half the pyrite was converted to the sulphate form.

The pressure leaching option would mean modifying the existing Elliot Lake plants by drastically altering the leaching operation, going to pressure leaching equipment (horizontal or vertical autoclaves, or pipe reactors) instead of air operated pachucas. The remainder of the plant would be similar to the present, although a somewhat finer grind may be necessary. Capacity of solid/liquid separating equipment would have to be assessed. A pyrite flotation step would have to be introduced on the leach tails to get a low pyrite tailing for disposal. The system has no advantages over the present approach so far as pyrite oxidation is concerned.

Additions to this work were the studies done at the Mines Branch by Dr. K. Downes and his staff (Ref 21, 22).

These studies talk about the conversion of pyrite to iron oxide and elemental sulphur by pressure leaching. The system really involves two steps. The first is the requirement of removing the labile sulphur atom from the pyrite by some pyrometallurgical step such as roasting:

$$2\ FeS_2 \xrightarrow{heat} 2\ FeS + S_2$$

The resulting pyrrhotite (FeS) can be pressure leached to produce elemental sulphur. The sulphur can then be removed by flotation and heat treatment. The result is a high conversion of the pyrite in the ore to marketable elemental sulphur and iron oxide. (Ref 24, 25).

The initial removal of the labile sulphur is a complicated and difficult pyrometallurgical step. It has not been resolved commercially in spite of considerable intensive study, and the operation of an industrial plant using the process (Barney Morrison, Noranda - personal communication). It is highly endothermic, which means that pyrite flotation would be required before the process could be applied. This means that so far as pyrite oxidation is concerned, the system is a flotation approach. The many complexities of the flowsheet, as well as the poor market outlook for sulphur, make this a non-option for sulphide removal.

The best depyritization option still seems to be the use of flotation for removal of the bulk of the pyrite, followed by some other approach to immobilize the small remaining amount of sulphide present.

6.6.2 Preconcentration

Preconcentration of Elliot Lake ore has been considered from the beginning of the camp. The methods used span the gamut of possible approaches: radiometric and photometric sorting, heavy media separation, magnetic separation and flotation. Recently two methods of preconcentration have had some success.

CANMET has shown that pyrite flotation, followed by high strength gaussian field separation will preconcentrate the sulphides and the radionuclides. Weight recoveries in the 30% range and reject tailings running about 0.1% S have been achieved.

Researchers at McMaster have emphasized flotation. Using a normal xanthate float, followed by a cupferron float similar recoveries and sulphide tailings have been obtained. A major problem with the McMaster system is the high usage and the high unit cost of cupferron.

The major benefit of any preconcentration system is a reduction in the size of the downstream chemical treatment plant, and the major disadvantage is the loss of uranium. Because treatment plants have all now been constructed, the preconcentration approach has no further practical application to the operation. Even at the low pyrite level, it seems likely that still further measures would be needed to restrain pyrite oxidation. Proportioning the acid production as was done in Section 5.4, a generation of about 170 tonnes/year of 100% acid still seems likely from a 400 hectare tailings area. While preconcentration will reduce the level of pyrite below that obtained with xanthate flotation, it is not a practical alternative for pyrite oxidation control.

6.6.3 Heap Leaching

Using the acid production of the thio bacillus usefully to remove uranium from the ore by heap leaching has been examined carefully in Elliot Lake by both producers, using the Agnew Lake experience as a guide. Tests have been done at CANMET and BCRC on the concept.

Whole ore leaching would involve crushing the ore to all minus 13 mm, and storing the ore for a prolonged period - up to 3 years - on a heap leach pad. During this time, the ore would be irrigated (and probably aerated) to maximize uranium leaching and pyrite breakdown. At the completion of the leaching the ore would be removed to a permanent disposal site.

The heap leach system has a number of advantages: (i) a much smaller surface plant, since only liquor would be treated, (ii) no problem with "slides" from tailings because they are so coarse, (iii) a considerable diminution of pyrite levels, since pyrite is the source of the lixiviant.

The system also has some major disadvantages: (i) the material must be moved twice, (ii) assured contact of the lixiviant with all parts of the pile isn't possible, (iii) extractions are excessively prolonged, and (iv) winter operation is difficult. Substantial pyrite still remains at the completion of the leach, so the pyrite problem still exists. Now that it appears likely all the Elliot Lake concentrators have been constructed, the total ore heap leach system does not seem a practical approach. The major benefit was a substantially reduced capital cost for the processing plant.

Denison is currently practising an underground heap leach on lower grade ore, and believe that eventually up to 20% of their uranium production may come from this system. Denison is to be commended on this approach. It will be a substantial "practical" benefit so far as reduction of pyrite oxidation difficulties are concerned, since it will mean relatively less sulphide tailings on surface for each kilogram of uranium produced. Presumably the Denison mine will be flooded on close down, and the low oxygen levels in the water, and the low transfer coefficients of oxygen into the pyrite will effectively quench any oxidation. The pyrite remaining in the mine will become innocuous.

6.6.4 Chloride leaching systems

Chloride leach systems have been mooted as possible ways to solubilize both uranium and the radionuclides, thus providing a virtually non-polluting final tailing. The work has been done by CANMET on two systems. McMaster University has developed a third rather interesting approach and uniquely turned this approach into a design project for fourth year chemical engineering students.

One approach by CANMET was examining gaseous chlorination of the whole ore. Quite preliminary work was done on this system. The approach necessitates that the ore be dry and be treated with chlorine gas in a fluidized bed. It is an impractical approach for a relatively low grade uranium ore being treated at high tonnage rates.

Hydrochloric acid was used as a lixiviant in one series of CANMET tests. It was noted in these tests that sulphides (as well as radionuclides) were a major environmental problem, and would have to be removed by flotation to obtain a relatively innocuous tailings. The sulphide was not removed previous to leach. It appears during the tests that oxidation of anywhere from 2.5% to 15% of the pyrite occurred, depending on the hydrochloric acid addition level and the oxidant used (nitrogen blanket, air, $NaClO_3$ and oxygen in increasing sulphide oxidation power).

While good uranium extractions could be obtained, extractions of thorium and radium required two stage leaching to be reduced to low levels. (Ref 30)

The second stage of leaching was either with HCl or with aqueous chlorine. Using aqueous chlorine initially followed by HCl was also examined as an option. Probably more sulphate was oxidized with the hypochlorite, but no readings are given. (Ref 29, 31)

The use of HCl and Cl_2 was primarily to explore radionuclide solubilization. Because a significant amount of sulphur was mobilized as the sulphate some reduction in pyrite occurred in these leaches. Only a very cursory examination was made of the possible circuit.

The methodology did not address, and was not meant to address, amelioration of the pyrite oxidation problem.

Fourth year chemical engineering students at McMaster University over several years have examined the development of "non-polluting" uranium flowsheets. The innovative flowsheets are based on studies done at McMaster under an NSERC grant. One approach examined was using ferric chloride leaching. This study incorporated an examination of the pyrite problem by considering flotation of the pyrite, and roasting the concentrate to produce acid which could be sold or neutralized. The overall study examined the difficulties of total flowsheet design and found several parts of the flowsheet where very tenuous process assumptions, with no experimental backing, were necessary to complete design.

None of the chloride processing options provide well developed alternatives to the current sulphuric acid leaching method, and none have any "practical" significance for pyrite oxidation.

6.6.5 Nitrate Leaching Systems

Nitric acid systems have been suggested for radium removal, and tested for this possibility by the Oak Ridge National Laboratory in the U. S. (Ref 61). A nitric acid leach is being used at Palabora in South Africa to dissolve uranium from a gravity concentrate made from copper tailings previous to recovery of the uranium using TBP solvent extraction. (Ref 63)

Additional leaching and conversion of the sulphide might be expected with nitric acid, but under usual conditions, the amount converted is propably small. In Elliot Lake, return of the nitrate solutions to the high temperature leach at Quirke and Denison gives little change in the sulphides.

Some work has been done on relatively low levels of ferric nitrate for radionuclide and uranium solubility. The data has not been reviewed, but it is believed that the problem of sulphides was not specifically addressed.

Little data is available on nitrate systems. It does not appear likely they offer any advantages over the current sulphuric acid approach.

7.0 PRACTICAL WAYS TO REDUCE PYRITE OXIDATION

This section examines those alternatives suggested as possible "practical" approaches to alleviate the problem of acidification of uranium tailings.

The requirements for acid production from pyritic uranium tailings are:

1. - presence of sulphide sulphur at more than a minimum level.
2. - presence of air (or oxygen). This is required whether abiotic or biologically catalyzed oxidation is considered.
3. - presence of water.
4. - presence of bacteria (for rapid oxidation to occur).
5. - proper temperature (5 C to 35 C).

For biological oxidation, nutrients such as CO_2, nitrogen and potassium are needed. The absence of calcite and probably phosphate rock also seems necessary.

Pyrite oxidation cannot be considered in isolation from the other specific problem of uranium tailings, viz: the presence of radionuclides. This is considered, although not exhaustively in the discussion below.

The possible approaches include:

1. - use of muskeg.
2. - flotation of pyrite from the tailings and subsequent disposal of the pyrite concentrate.
3. - deep water disposal.

4. - limestone addition.
5. - phosphate rock additions.
6. - sodium lauryl sulphate applications.
7. - flotation plus pressure leaching.
8 - tailings flotation and acid production.
9. - use of impervious membrane plus cover.

All of these approaches could be considered at current active tailings sites. Items 1,4,5,6,8 and 9 might have application at inactive tailings areas.

7.1 MUSKEG

Dr. Marvin Silver at CANMET has done considerable work on Elliot Lake pyritic uranium tailings. Much of this work has been done in lysimeter towers, packed with Elliot Lake tailings. The objective of the work has been to provide a predictive mechanism for determining what will happen in the tailings on abandonment.

In one series of tests, Dr. Silver tried the effect of a thin layer of muskeg on the tailings. These particular tailings were generating acid. A 4 cm layer had little effect but a 30 cm layer turned the system around and stopped acid generation (Ref 28).

Muskeg is a mixture of live mosses and dead plant material found in the Canadian Shield generally in wet swampy areas. It may have the potential to stop acid generation from old tailings areas. (Ref 47)

The Sherman Mine just outside North Bay is an iron mine, which treats occassional ore high in pyrite (Bob Garrett-Sherman Mine - Safety & Environment Administrator - personal communication). Sherman has tried a 2 foot layer of muskeg over the top of pyritic fill in an open pit. The unprotected muskeg quickly oxidized, and any benefit was rapidly lost.

In order for this approach to be "practical" the potential tailings area should be low-lying, and in a topographic environment which would allow it to be converted to a "bog". Some areas in Elliot Lake seem to have that possible potential.

The live plants in muskeg thrive in an acidic environment, about pH 4.5, but they can be killed by pH levels which are either too high or too low. The plants require high water levels, as they are 70% to 90% water. The living plants need sunlight, air, and water to grow. They use oxygen and fix nitrogen in the top layers, and continue to build up residual material. The water content, and the growing mat of dead fibre should prevent penetration of oxygen to the tailings solids. (Dr. Marvin Silver - personal communication).

The testing requirements would be to isolate a wet section of tailings so this portion is not affected by run-off, This portion of the tailings could be treated to stop bacterial action using SLS or applying limestone. It is critical to keep the pore water in the plant at the right acidity level. The chosen test plot would be covered with muskeg harvested from swamps in the area. A couple of years growth should determine if the muskeg would flourish, or whether it will die out. Meantime an assessment of the requirements for engineering, the availability of muskeg in the area, and an assessment of harvesting and planting labour could be completed. Monitoring of effluent from the muskeg covered area would also be required to assure this method stopped acid generation.

The method has promise because it would provide a self perpetuating cover using a natural material indigeneous to the area and sourced close to the tailings themselves. A disadvantage is that the method is not generally applicable, being useful only for tailings with a specific topography.

7.2 FLOTATION

One possible approach to the problem of acid generation which some people have recommended is flotation of the pyrite from the tailings and storage of this pyrite deep within the tailings pond. This method was suggested by CANMET as part of a larger suggestion to include flotation of radionuclides. The CANMET pilot plant work on Elliot Lake tailings was tried on a large scale at Denison Mines and reported on by T. Eric Pakkala, metallurgist at Denison. (Ref 27). While the pyrite flotation showed good sulphide removal, the radionuclide removal could not be construed as satisfactory.

As an example of how the system might be applied, a hypothetical tailings area is considered. It is assumed that the plant mills 5000 t/d of ore for 350 days/year for 5 years previous to shutdown. The milling flowsheet is shown in Fig 7.7-3 which is taken from the Pakkala report. The tailings area is taken as 160 hectares.

The data on recovery and grades is taken directly but selectively from the Pakkala report (only 90% plus sulphide recovery runs were selected). Assuming a sulphide content of 3% for tailings during the period a concentrate and tailings tonnage and grade can be determined. With some experience, it is believed that the distribution of sulphide can be achieved or improved (e.g sulphide in tailings will generally be lower than the average postulated). (Table 7.2-1).

The operating methodology is assumed as follows:

1. The tailings are divided into five equal sections, each about 32 ha.

TABLE 7.2 - 1 FLOTATION OF PYRITE

RESULTS FROM DENISON PILOT PLANT RUNS

(From Report by Eric Pakkla of Denison)

RUN DATE (1978)	PAKKLA REPORT TABLE #	lb/ton of REAGENTS		SULPHIDE CONCENTRATE	
		COLLECTOR AMYL XANTHATE	FROTHER PINE OIL	WEIGHT %	S DIST'N %
Aug 29	5	0.19	0.64	10.7	96.5
Aug 31	7	0.33	2.29	8.1	93.8
Sept 4	8	0.42	2.45	8.8	90.8
Sept 5	9	0.33	1.98	7.9	91.9
Sept 9 *	10 *	0.26	2.48	7.0	94.7
Sept 7 *	11 *	0.22	2.02	8.0	91.3
Sept 8 *	12 *	0.20	1.5	9.7	95.3
Sept 12	14	0.20	0.7	7.4	90.7
Sept 14	16	0.19	0.8	11.4	91.5
Sept 18	18	0.16	1.07	15.4	92.0
Sept 21	21	0.14	0.61	11.7	91.0
Average 11 runs		0.24	1.50 **	9.6	92.7

NOTE: At 3% S in feed, the sulphide concentrate will average 29% S and amount to 480 tonnes/day while the 4 520 tonnes/day of low sulphide tailings will run 0.24 % S.

* Numbering & dating from actual report.

** Unusually high frother addition, perhaps due to salt content of solution or short contact time during flotation.

2. The flotation plant commences operation. The pyrite concentrate is pumped to the first of the five sections. The low sulphide tailing is stored in a separate and isolated area, capable of accepting 1.6 million tonnes of solids.
3. After 1 year the pyrite disposal is moved to the second area. During the previous year, some 168,000 tonnes of pyrite have been laid down over 320,000 m^2 of tailings. Assume a concentrate of solids specific gravity of 3.3, and solids separating to 85% solids. The specific volume of the pyritic tailings would be 0.4076 m^3/t. The pyrite would thus be essentially contained in a layer averaging 0.21 m thick (about 8 inches). Of course, because of the way tailings are laid down volumetrically, the layer will vary substantially from the average.
4. The pyritic material is covered during the second year with low pyritic material. This material will settle to about the same 85% solids density, but the specific gravity will be about 2.66. The specific volume of this material will be about 0.4695 cu m/t. The 1,582,000 tons of low sulphide tailings will provide a coverage about 2.28 m thick on average above the sulphur containing material (about 7.5 feet).
5. This same proceedure is carried on to the completion of the five year period. When the mine shuts down, active tailings production ceases. One section of the pond remains with a high pyritic cover. Mechanical equipment - trucks, front end loaders etc. - are used to reclaim the initial year's production of low sulphidic material and spread this finally over the last 32 ha.

An additional modification to this basic sulphide float suggestion would be to include limestone additions to the final low sulphide tailings. The 0.24% S would neutralize about 8 kg/t of 95% limestone if it all turned acid. So a limestone addition of 10-12 kg/t on low sulphur tailings would provide considerable protection against acid generation and is suggested as one of the reagents which would have to be added if the plan were to be implemented.

This proceedure results in a final cover of relatively low sulphide material. It does not address the radionuclide problem. In examining the flotation results in the Pakkala report it appears that it may be possible to concentrate as much as 40% of the radium in the sulphide concentrate if an activator is added, with some improvement simultaneously in pyrite recovery. This might result in a substantially lower radium level in the covering tails (reduction from about 360 pc/g to about 240 pc/g). To do this would increase the weight reporting to the sulphide concentrate, and thus reduce the thickness of the low pyrite cover and would require some additional reagent, such as oleic acid.

An additional consideration with this approach would be finally contouring the tailings to provide a slight slope, sufficient to remove precipitation rather than allowing it to remain on the surface and percolate through the mass of tailings. Vegetation of the surface would also be needed to control dust and improve esthetics.

The additional capital required for installation of this system would include the following:

1. An added flotation plant capable of treating 5000 t/d of tailings from the last stage of solids washing. This would include all the equipment and the building housing the plant.

2. A prepared area for lay down of the low sulphide tailings.
3. Arrangement for a second tailings disposal pumping and spigotting system.
4. Required equipment for "extra" reagent feeding. (e.g. limestone, possibly fertilizers and organic material for the top layers if one wished to vegetate.)

The additional operating costs would include:

1. Additional cost for heating the new plant, and for electricity and water.
2. Labour and supplies for the flotation plant operation.
3. Flotation reagents (average)
    - (a) 0.12 kg/t or 210 t/y of potassium xanthate.
    - (b) 0.75 kg/t or 1313 t/y of pine oil.
    - (c) 12 kg/t of ground 95% pure limestone. (21,800 t/y).

Note that the low reagent costs from the Pakkala testwork were 0.07 kg/t xanthate and 0.32 kg/t of pine oil.

If one attempted to float some radionuclides by adding oleic acid, additional reagent costs might include 1 kg/t of oleic acid. Other materials such as peat and fertilizers would further add to these costs.

Just the reagent costs of pyrite flotation would add substantially to the operating costs at the mines. Such additional expenditures may, however, be justified if they resulted in a final lower treatment cost at mine shutdown.

In order to better define the benefit of flotation, R.V. Nicholson has applied the experimental data developed at the University of Waterloo (Ref 26). He has calculated the rate of oxidation of pyrite and the acid generation in the tailings (see Appendix D).

The parameters used are:

- overburden of low sulphur float tailings running 0.24% sulphur as sulphide in pyrite
- depth of overburden 2.3 m
- limestone content of overburden 12 kg/t of 95% pure calcite, ground all passing 44 microns
- layer beneath overburden layer is 29.0% S as sulphide in pyrite, and is 0.2 m thick, running from 2.3 m to 2.5 m from surface of tailings
- below this, from 2.5 m down, would be 3% S content tailings.

Mr Nicholson has calculated that the limestone would prevent pyrite oxidation in a time frame which could range from 30 years to 420 years. The assumptions consider the flux of carbonate and oxygen into tailings and are given in the appendix. (Appendix D).

The required testwork to define the efficiency of flotation has already been done. The removal of most of the pyrite would not, however, result automatically in a stable cover, not producing acid. It would make pyrite oxidation control substantially easier. Oxidation control would require acidity release monitoring, with possibly very occasional remedial action. This system would be a "practical" way to substantially change the kinetics of pyrite oxidation and the potential amount of acid and heavy metal salts which can be generated. The calculations shown in section 5.4 also indicate testwork is needed to determine how the oxidation rate varies with the sulphide content. Mineralogical studies are also required to see how the remaining sulphide is locked in the mineral lattice.

7.3 UNDERWATER DISPOSAL

A system of deep water disposal might be a suitable approach for preventing the oxidation of pyrite in pyritic uranium tailings. Specifically, in Elliot Lake, the method might be a practical approach for the mines on the shore of Quirke Lake as the current tailings basins approach capacity.

A study of deep water tailings disposal was initiated by Rio Algom and Denison mines in December 1980, and a comprehensive report examining this option was produced by SENES Consultants Ltd. and Golder Associates in October of 1981. (Ref 9). This report was the first stage of a continuing study of deep water disposal, but is still in the "draft" stage.

The report examined the physical parameters which might preclude deep lake disposal, for instance the effect of added pressure from deposited tailings on underground workings below the lake. This did not constitute an unacceptable risk. The report also considers the system transport and underwater release methods which have been used elsewhere, and discusses their application for Quirke Lake. The SENES-Golder report examines nine separate scenarios. These include three land-based scenarios, and six underwater scenarios. The land-based options consider mill expansions and process variations. The underwater scenarios consider these options and various fresh water diversion and treatment options. All these scenarios are modelled to examine ammonia and radium problems and T.D.S.

It is not explicitly defined how the authors envision treating the tailings areas which are on surface at the shutdown. Apparently some undefined method of preventing acid emissions was assumed.

As an example of these land based emissions, the input data for the Quirke surface disposal tailings basin shows the following (Ref 9, Scenario 1, approximate averages):

MATERIAL	TIME PERIODS - MONTHS		
	0-180	180-240	240-708
TDS (mg/L)	2700	2100	1000
NH_3 (mg/L)	30	2	0.2
Ra (pCi/L)	7	3	125

This appears to indicate a cessation of lime and barium chloride treatment 5 years after closedown, and no acid generation after this time. Much longer treatment for acid run-off will probably be needed, based on the background from Nordic (see Section 5). The surface impact requirements defined by the model therefore appear underestimated if all treatment ceases 5 years after closedown.

The practibility of the deep water disposal system requires the following:

(a) that the disposal method not require diversion of the fresh water flows from Quirke Lake, nor treatment of Quirke Lake outfall.

(b) that the current level of "contaminants" in Quirke Lake be maintained or even reduced.

(c) that deep water disposal be clearly demonstrated not to have a major negative environmental impact.

(d) the process changes required to implement the system be reasonably small and readily manageable.

These items are considered below.

Once the tailings were settled underwater, the oxidation of pyrite should be negligibly slow. This seems to be defined by the experience with deep lake disposal at other places mentioned in the SENES report. If pyrite pollution only is considered, deep water disposal is practical and probably the best approach of all. (Ref 38).

The other materials in the tailings are the problem. Levels of ammonia, radium, and T.D.S. are suggested as rising in Quirke Lake by the model presented. One way to eliminate the ammonia problem is illustrated in the attached drawing. Only the washed solid tailings go to Quirke Lake. The other soluble materials would go to the present discharge areas. (see Drawing NUTP H-01).

The generation of a high T.D.S. level because of dissolution of gypsum may be attacked in two ways. If cold Quirke Lake water were used to slurry the tailings at a reasonably high density before the solids were transported to the lake, probably gypsum dissolution would be reasonably slow. Tests are needed to determine this. If gypsum solubility were still a problem it could be eliminated entirely if the leach discharge rather than the partial neutralization discharge were filtered. This is cetainly more difficult operationally. However, both Denison and Quirke filtered leach discharge directly on the disk filters at the begining of Elliot Lake operations in the late 50's, and Milliken did the same in rotary drums in the early 60's. So the approach is not "impractical" if this were all that was needed.

The major unknown is the behaviour of radium. In the flowsheet, this is shown as washing out. This element, because it is there in such minute amounts, behaves unpredictably. There is no assurance radium dissolution could be suppressed until the tailings settled underwater and test work is required. Perhaps barium hydroxide or barium carbonate, added directly to the filtered solid tails might help. The model assumed the solution entering Quirke Lake with the solid tailings contained 1250 pCi/L of dissolved radium, and that substantial additional radium dissolved. This led to radium being a problem.

With the possible (and important) exception of radium, then, both items (a) and (b) can be satisfied. Item (d) seems to be satisfied. Are there tailings deposits now under water which could be used to assess long range environmental impact (Item (c))? Such tailings deposits exist for base metal sulphides. (Ref 9). At Elliot Lake, the slide of Stanrock tailings into Quirke Lake may aid in providing an answer. Tailings sampling of Westmin base metal sulphide tailings under Buttle Lake on Vancouver Island was done by B. C. R. C. personnel. (Laurie Hunt - BCRC - personal communication - (604)224-4331) Sampling of the Stanrock underwater tailings by similar methods could determine the constituents of the pore water, and may allow a better model to be developed on the effects of underwater disposal. This is particularly important for radium levels.

So far as pyrite is concerned, the deep water disposal scenario can be considered a close approach to the "no pollution ever" option mentioned at the beginning. It is the only approach considered herein which does not seem likely to require perpetual care of some type. Certainly in the range of 100 years, this appears the most advantageous approach. If the small process change suggested would mean the lengthening of surface tailings basin life, then this option may also have substantial economic benefit where it can be reasonably applied.

The major disadvantage of this option is that if it did lead to significant pollution, remedial methods are more difficult. Stream diversion and treating Quirke Lake outfall are still, of course, alternatives. Another approach might be grinding neutral rock, and laying this down over the underwater sulphide deposits.

In summary, it is believed that a reasonable and practical deep water disposal system can be developed for some pyritic containing Elliot Lake uranium tailings. The key to the system is separating the solution from the solids in the mill, with very minor changes to the existing equipment and operation. The suggested approach would not require either the diversion of fresh water flows, or extra treatment facilities. Nor would it change current pollutant levels in the lake.

Proposed testwork would directly examine an existing tailings deposit within the lake. As well, the efficacy of the filtering to remove radium and the rapidity of gypsum dissolution from the solids should be determined.

7.4 LIMESTONE

Acid run-off from the tailings area can be readily neutralized with slaked lime. Lime neutralization is efficient and effective in neutralizing acid and acid salts. The disadvantages of lime are: (i) it is a strongly caustic material which can easily increase the water pH levels and cause damage to biota (ii) it does not produce a dense precipitate on reacting with acidic run-offs (iii) it is significantly higher in cost than limestone.

Limestone is a natural product which will neutralize acid waters to a pH 7 or above. It's advantages are: (i) it can't over-neutralize - the maximum pH achievable with limestone is about 8.0 (ii) it produces a dense precipitate which is easy to settle and filter (iii) it is substantially lower in cost than lime - approximately half the cost of quicklime for equivalent neutralizing power.

Limestone has some disadvantages: (i) to be effective limestone has to be finely ground - according to some U.S. workers in the 5 to 10 micrometer range (Ref 19) (ii) the oxidation of ferrous iron to ferric is relatively slow so oxidation retention times must be longer than with lime (Ref 19). If the current neutralization technique is to be kept operating, limestone treatment should be considered as a potential replacement for lime in neutralizing tailings run-off.

Because it is inexpensive, limestone could also be considered as an additive to restrain bacterial action. The addition of ground limestone in the top layer of the tailings, even without flotation separation, should mitigate bacterial action. The length of time of suppression will depend on the amount of limestone added. The effect of only a small quantity of limestone was defined in a test series by Jongejan (Ref 37). In this reference, Mining Company #8 is believed to be from Elliot Lake. The 0.5% level of carbonates (5 kg/t) proved to be too low to control the acid releases with a sulphide content running near 5%. If limestone is to be used as a buffer, then, on regular tailings, quantities substantially greater than 5 kg/t (10 lb/short ton) may be needed.

Two series of tests are suggested to evaluate the applicability of limestone. The first series would employ limestone ground to the fineness now used for partial neutralization at Quirke (all - 325 mesh) as an alternative neutralizing agent to lime in existing treatment lants for inactive tailings run-off. These tests would indicate how quickly ground limestone will neutralize the acid and precipitate the iron in effluents, at ambient temperatures. The excess needed to get to pH levels of 6.0, 6.5, 7.0, and 7.5 would be determined. The contact time needed to bring down the iron and acid would also be determined. At the end of the test series, the cost effectiveness of limestone as compared to lime in run-off neutralization could be determined.

The second test series would determine the buffering action of finely ground limestone (all - 325 mesh) intimately mixed with wet tailings. Several addition levels, starting at 5 kg/t should be used. The test program could use 45 US gallon plastic pails with porous bottom plates (See Figure 7.4 - 1, after Ref 46). Accelerated tests using high wash rates, such as were used in the lysimeter tests (Ref 28) and in Jongejan's work (Ref 37)

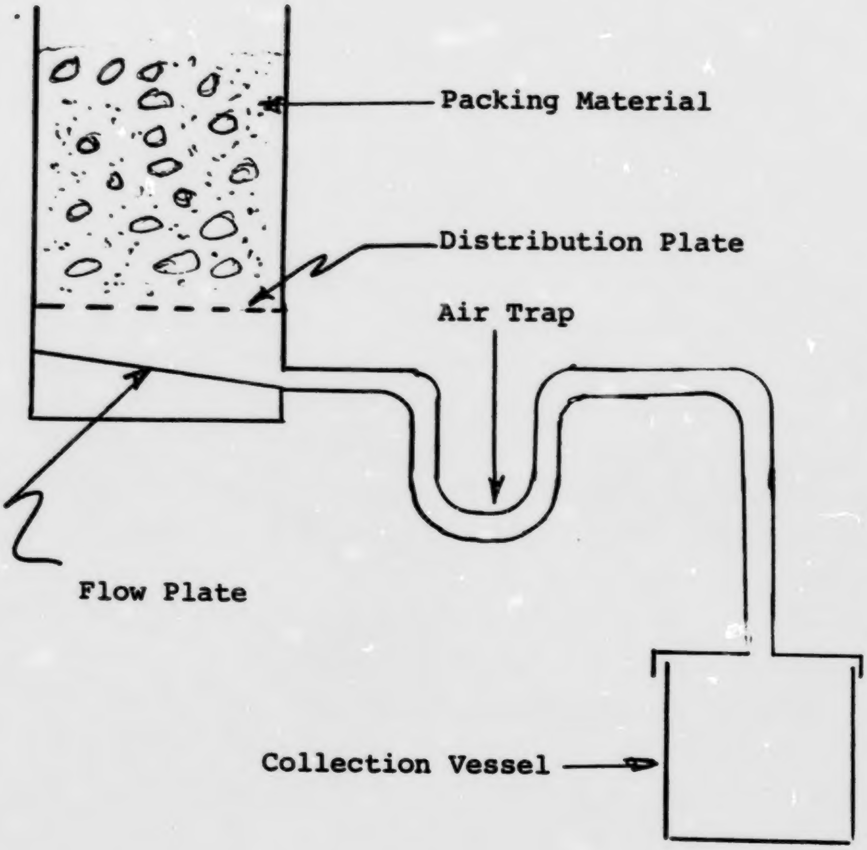

Packing Material
Distribution Plate
Air Trap
Flow Plate
Collection Vessel

Figure 7.4 - 1

SIMPLE LYSIMETER SYSTEM

(After Renton & Stiller, Ref 46)

could be run in the laboratory. Outside test programs using similar equipment, but subject to natural conditions (freezing and thawing, natural precipitation) would be run as a comparison. Sulphate, carbonate, iron, calcium, and pH would be analyzed in the collected effluents.

Waterloo graduate students (Ref 26), working under Dr. John Cherry, are exploring the kinetics of pyrite oxidation, and particularly the effect of carbonate ion on those kinetics. (See Appendix D). Their work should aid in modelling the limestone addition levels needed to keep pyrite oxidation rates low. Data from the second test program should be used to validate the information coming from the Waterloo studies.

One other approach might be studied. Researchers in the U. S. have used limestone percolation beds, and Rio Algom did some work on this system at Milliken some years ago. A substantial static head is required. This provides the requisite energy to move the stream through the limestone bed. The objective is that the oxidation products, the iron hydroxides and the gypsum, be removed by the flow, so a fresh limestone surface is always being produced to react with the acidic waters. The elevation of the top of the tailings basins should provide the requisite static haed. Even if complete control of acidity is not achieved, the reduced levels of acid and iron may mean Ms Kalin's suggestion (pp 6.6 ff) of a "vegetative area" could now be used for a final polish. Further work is needed on this approach, particularly as it may reduce present operating costs.

7.5 PHOSPHATE ROCK

Phosphates have been suggested as a method of controlling radioactive emissions in uranium tailings (Ref 35). The same attributes which make phosphates a possible restriction for radioactivity - that is, their very low solubility - may also be useful in pyrite oxidation control. A group of professors in the Chemical Engineering Department of the University of West Virginia have been testing phosphate rock for the latter purpose.

According to Dr. Alfred Stiller (personal communication) the key to the control is the reaction:
$7\ Fe_2(SO4)_3 + FeS_2 + H_2O \longrightarrow 15\ FeSO_4 + 8H_2SO_4$.
It is this critical reaction which determines the rate of pyrite oxidation (Ref 20). The bacteria oxidize ferrous iron in solution to ferric, but the oxidation of pyrite cannot occur if the ferric ion is removed from solution. That is the function of the phosphate. A small quantity of phosphate will tie up the ferric ion and prevent further oxidation from occuring. (Ref 58)

A practical application of the method has been developed in West Virginia. (Dr. Stiller - personal communication). The overlying pyritic strata above the coal seams are removed and placed aside. The coal is then stripped, and the pyritic material backfilled into the pit. The backfilled material is sprayed with finely divided phosphate. The addition is about 0.1% by weight compared to the waste; this level is said to result in "self control" of the pyritic oxidation. The waste is then covered with the original top soil which had been set aside during the mining, and the area is vegetated.

Dr. Stiller also noted that work done by his group shows that phosphates will aid in the coagulation of the wastes produced by liming of acid mine drainage, giving a much better settling and filtering product.

The use of phosphate rock to control acid mine drainage seems to be relatively recent. Several papers (Ref 48, 50, 51) show considerable interest is being generated by the suggestion. Tests done by an operating company (Ref 48) leads to the conclusion that about 0.3% by weight of phosphate rock will control and effectively prevent the release of acidic mine drainages. The author speculates that this level will act as a control for even high sulphide materials, because sufficient phosphate ion is generated to control the ferric iron as soon as it is formed.

In Elliot Lake, it would be relatively simple to move phosphate rock into the area by boat from Florida up the Great Lakes. Supply would probably be simpler than in West Virginia. The material could be ground in the concentrators, and added directly to the tailings. If a level of 0.3% (3 kg/t) were effective, the method should not be excessively expensive.

Test work needed to develop the concept would inculde the following:

1. Laboratory tests on the tailings neutralization circuit. These tests should adjust the pH of the circuit with ground limestone to a pH of about 4.0 to 4.5. Limestone should be relatively effective in raising the pH to this level. The time taken to oxidize the iron all to ferric, which should precipitate at this pH, should be determined. This should be compared to the kinetics with slaked lime. Finely ground phosphate

rock should not be added until the acidity has been neutralized to at least 4.0, and all the iron is down as ferric hydroxide. (The lowest solubility product for the ferric phosphate complex, $FePO_4$, is 5.5 pH. Phosphate rock is acid soluble, and the level of phosphate in solution rises substantially at a pH below 4.0). Additions of 0.5, 1.0, 2.0, 5.0, and 7.5 kg/t of rock could be tried. This will bracket the 3 kg/t level which is suggested as optimum.

2. The resulting neutralized slurry should be settled in small lysimeters, and thoroughly washed to remove all the soluble salts. Washing with fresh (Dunlop Lake) water should be continued, examining the levels of TDS, pH, iron, and radium in the effluent at intervals. Data should be compared with CANMET or CCIW lysimeter tests.

3. If no rapid oxidation of pyrite occurs, larger tests outside, in something similar to the Rio Algom test pits, will be required.

4. If the method proves promising, an economic study would be needed to examine the cost of moving phosphate rock to Elliot Lake, storing the necessary quantity for winter supply, transport to the mines, and the requisite preparation for addition to the tailings neutralization circuit. Until the efficacy of the method is proven, adequate data is not available to prepare a reasonable cost assessment.

7.6 SODIUM LAURYL SULPHATE

The U.S. Bureau of Mines has studied the use of anionic detergents to inhibit or kill thio bacillus. (Dr. Robert Kleinmann - personal communication). Several papers have been presented on the approach (Ref 18, 19).

The authors have extensively studied the reaction mechanism for pyrite oxidation (Ref 20). They consider this to occur in three phases (See Table 5.2 - 2).

The first phase proceeds at a pH of above 4.5 and includes abiotic pyrite oxidation to ferrous sulphate, and ferrous iron oxidation to ferric hydroxide. These same reactions proceed at pH levels of 4.5 down to 2.5 in the second stage, with bacterial action catalyzing the oxidation of the ferric iron. This step was abiotic in the first stage. The third stage, below 2.5, consists of ferric iron oxidation of pyrite forming ferrous iron, and oxidation of this ferrous iron to ferric iron in solution by bacteria, This latter reaction is the rate controlling step. Reducing the rate of oxidation is the objective.

The bactericide action of anionic detergents increases at lower pH levels. At levels of 10 mg/L and a pH about 2.5, the detergent sodium lauryl sulphate (SLS) appears to break down the fatty tissues in the cell wall, causing disaggregation and effectively killing the bacillus. SLS detergent is biodegradable over time, so has no long term impact. The main problem is applying the detergent to get best bactericidial action.

Application methods include spraying an SLS solution over the tailings area. A typical application rate would be 55 US gallons per acre of 30% solution, diluted 175:1.

This kind of treatment will substantially decrease the oxidation rate, and is said to cost about $200 US per acre. This treatment must be completed every four months or so to be effective.

An alternative treatment is incorporation of SLS into rubber formulations which release the chemical slowly over time. This method may provide a protective bactericide availability for up to 3 years. Tests on this approach are currently (April/84) continuing.

For large quantities of inactive tailings, the use of a bactericide is not a "practical" approach. However, if a quenching of bacterial activity is required at an inactive tailings site because some remedial action is to be instituted, then the method might have significance.

Dr. R. Kleinmann, who is the supervisory geologist at the U.S. Bureau of Mines station doing this work, has volunteered to try the method on samples of old Elliot Lake tailings which are generating acid. Such a test program would consist of shipping 10 to 12 large 45 US gallon drums containing samples of tailings to the US facility. Smaller samples would be difficult to properly assess. The samples would be treated with varying levels of SLS. On completion of the tests, the operators in Elliot Lake would know the amount of SLS needed, and be better able to decide if the process has practicality.

The best way to send samples would be through CANMET because there is an information exchange arrangement between CANMET and the US Bureau of Mines. The pertinent address:

US Bureau of Mines,
Pittsburgh Research Centre,
P.O. Box 18070,
Pittsburgh, PA. 15236.
Attention: Dr. Robert Kleinmann or Patricia Erickson.
Phone: (412) 675-6555.

Either Dr. Kleinmann or Patricia Erickson can be contacted to discuss the research program in more detail.

7.7 FLOTATION PLUS PRESSURE LEACHING

If some immediate process benefit could be obtained from pyrite flotation, the approach has more chance to be considered "practical". One possible approach, covered by a patent (Ref 23), is the flotation of a pyrite concentrate followed by pressure leaching of this concentrate to produce acid. (See Fig. 7.7.1).

Two approaches may be considered. The first would float the pyrite concentrate directly from the feed to the plant, before the leach circuit. The positive aspect of pre leach flotation is that the ore is clean, and reagent costs, particularly frother, will probably be much lower than for flotation after the tailings. Tests have been done in Ottawa on the Continuous Process Development Unit (CPDU) on Elliot Lake ore. (Ref 33) (Fig. 7.7.2). This is compared with the results from a pilot plant study on flotation run at Denison Mines and reported by E. Pakkala (Ref 26) (Fig. 7.7.3). These results are compared in Table 7.7.1. The much higher reagent consumptions in the Pakkala report are probably because of the high dissolved salts content in the solution feeding the float circuit.

TABLE 7.7.1
COMPARATIVE FLOTATION RESULTS

	CPDU		Denison	
	g/tonne	lb/ton	g/tonne	lb/ton
Frother -- Dowfroth	80	0.18		
-- Pine oil			680	1.50
Xanthate -- KAX			109	0.24
-- NaAX	100	0.22		
% of S Recovery	92.7		92.7	

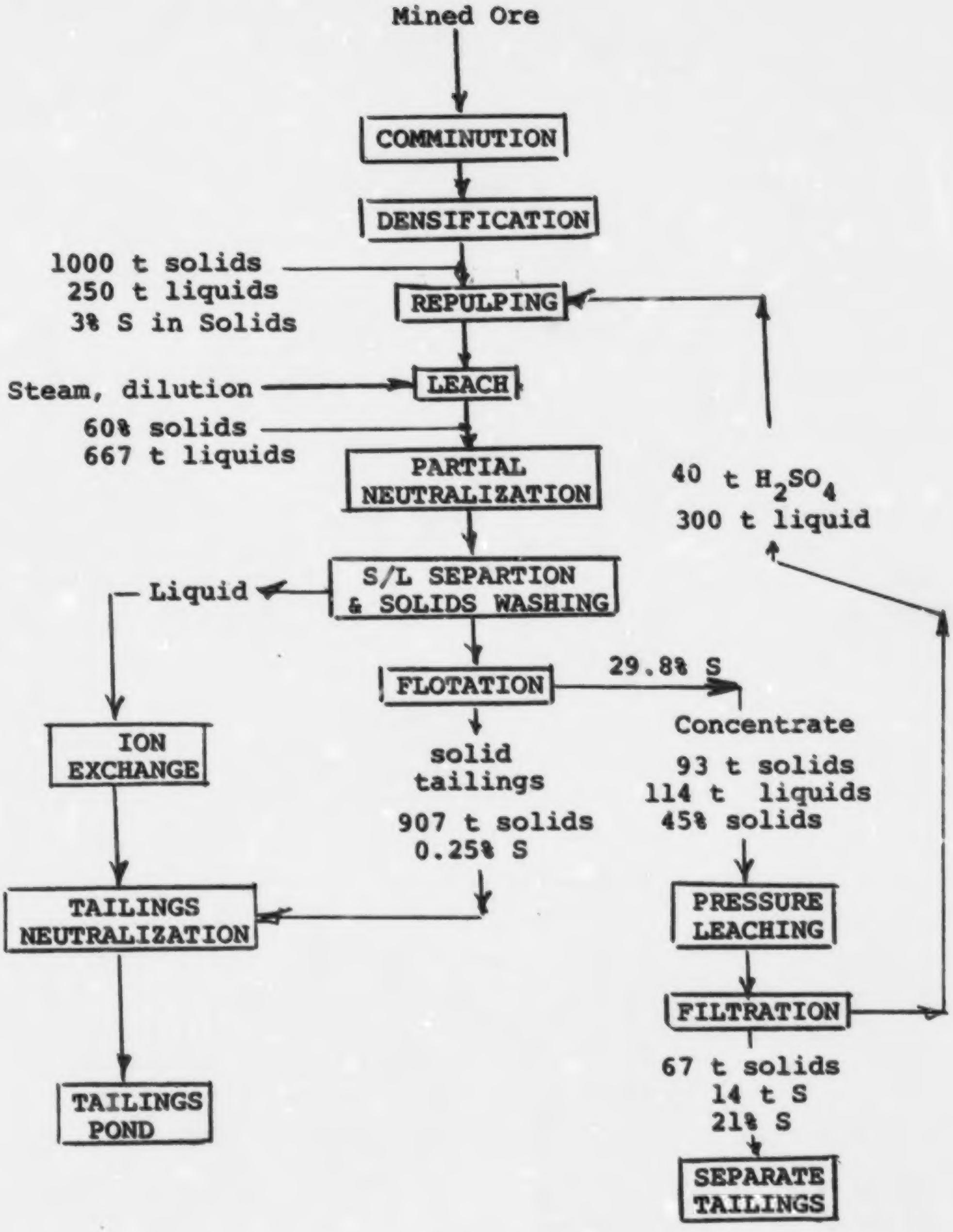

Mined Ore
COMMINUTION
DENSIFICATION
1000 t solids
250 t liquids
3% S in Solids
REPULPING
Steam, dilution
LEACH
60% solids
667 t liquids
PARTIAL NEUTRALIZATION
40 t H_2SO_4
300 t liquid
S/L SEPARTION & SOLIDS WASHING
Liquid
FLOTATION
29.8% S
Concentrate
93 t solids
114 t liquids
45% solids
ION EXCHANGE
solid tailings
907 t solids
0.25% S
TAILINGS NEUTRALIZATION
PRESSURE LEACHING
FILTRATION
67 t solids
14 t S
21% S
TAILINGS POND
SEPARATE TAILINGS

Figure 7.7.1
FLOWSHEET- CONCENTRATE PRESSURE LEACHING

Figure 7.7.2

Copied from Ref. 33

FLOTATION CIRCUITS IN C.P.D.U.

FIG. NO. 3

RUNS NO. I AND 2

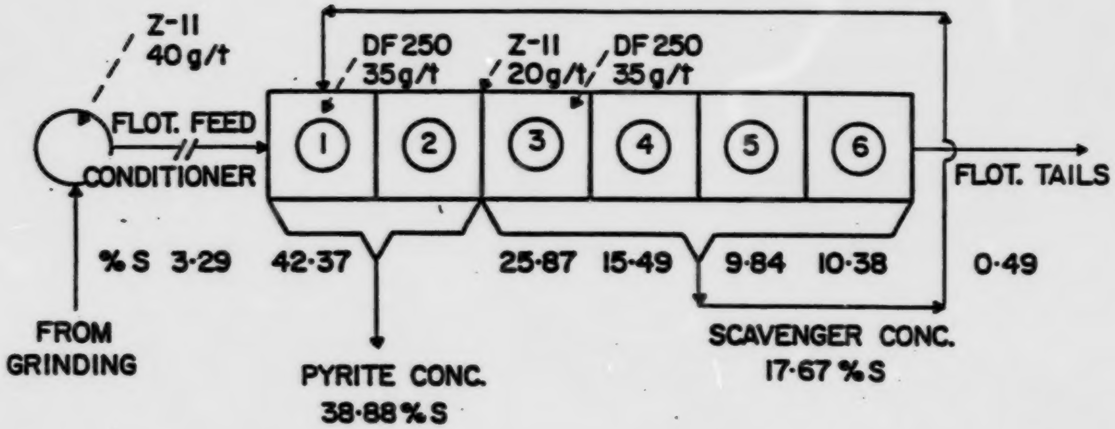
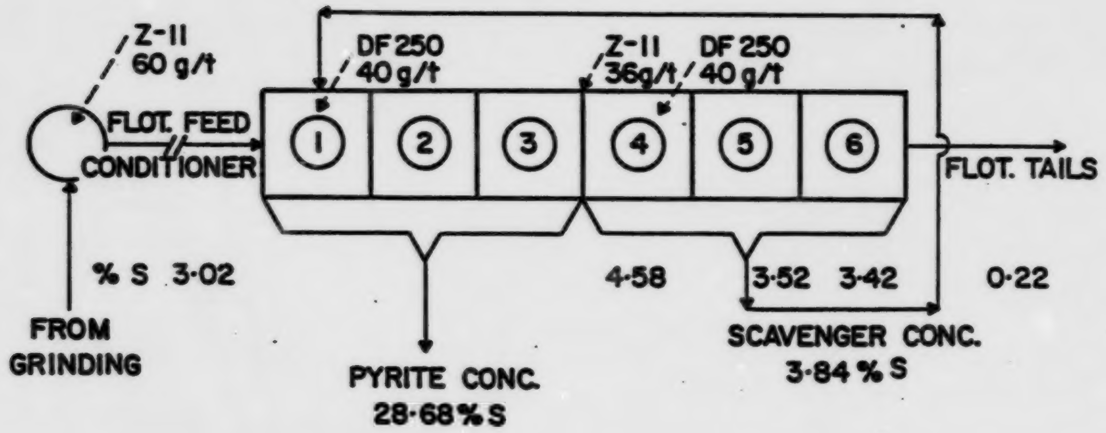

Z-11
40 g/t
DF 250
35g/t
Z-11
20g/t
DF 250
35g/t
FLOT. FEED
CONDITIONER
1
2
3
4
5
6
FLOT. TAILS
% S 3·29
42·37
25·87
15·49
9·84
10·38
0·49
FROM GRINDING
PYRITE CONC.
38·88 % S
SCAVENGER CONC.
17·67 % S

FIG. NO. 4

RUN NO. 3


Z-11
60 g/t
DF 250
40g/t
Z-11
36g/t
DF 250
40 g/t
FLOT. FEED
CONDITIONER
1
2
3
4
5
6
FLOT. TAILS
% S 3·02
4·58
3·52
3·42
0·22
FROM GRINDING
PYRITE CONC.
28·68 % S
SCAVENGER CONC.
3·84 % S

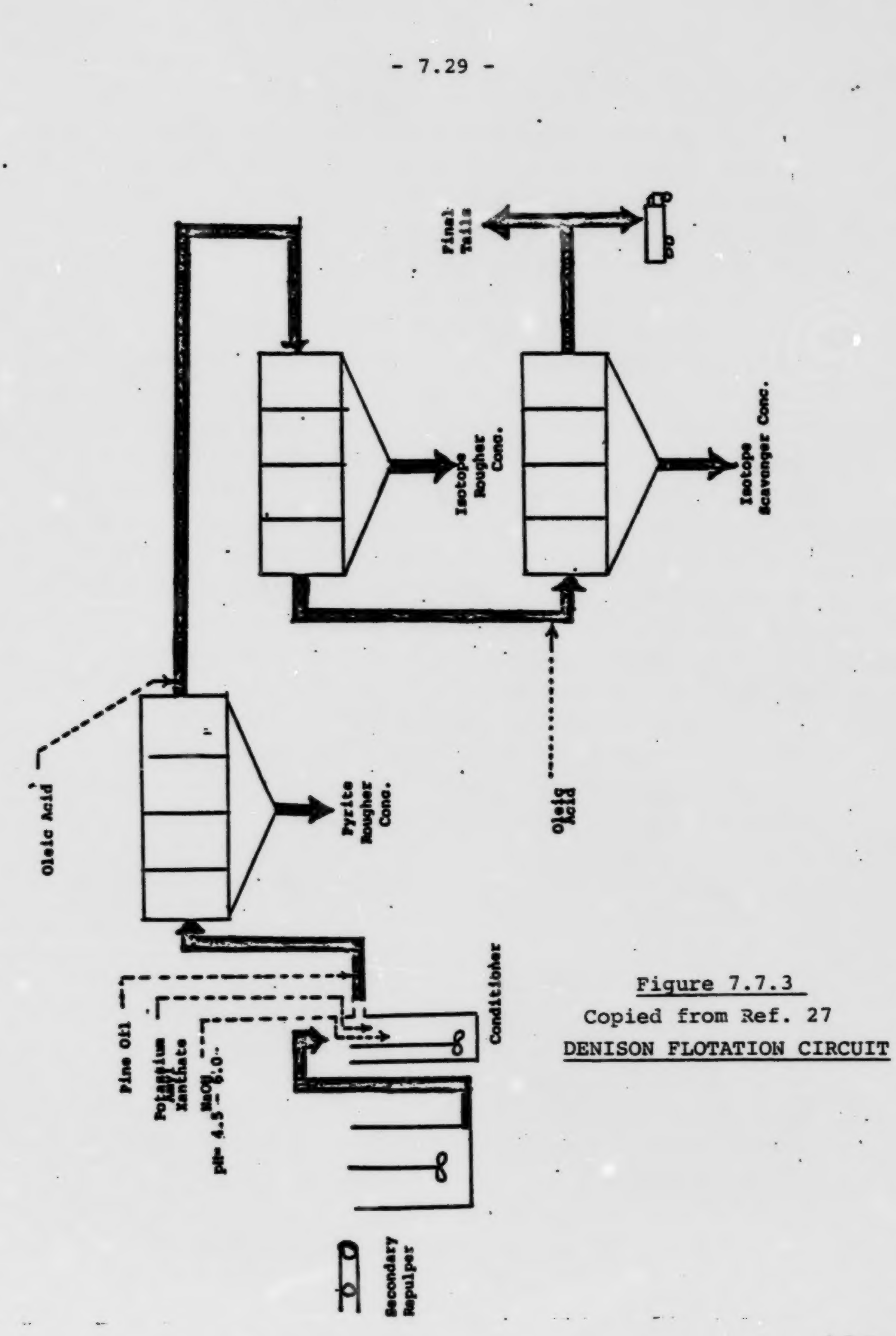

Final Tails
Isotope Rougher Conc.
Isotope Scavenger Conc.
Oleic Acid
Pyrite Rougher Conc.
Oleic Acid
Pine Oil
Potassium Amyl Xanthate
NaOH
pH= 4.5 - 6.0
Conditioner
Secondary Repulper

Figure 7.7.3
Copied from Ref. 27
DENISON FLOTATION CIRCUIT

While flotation appears simpler on the clean ore, the disadvantage is that the tailings must be thoroughly washed to recover the uranium before the next stage. On balance, it would appear that tailings flotation is the better approach and this is examined further.

A preliminary block flowsheet is given in Fig. 7.7.1. The normal comminution circuit is used. The ground solids are densified as high as possible, say to 80% solids. The filter cake is combined in a repulper with filtrate from the pressure leach. The partial neutralization and solid liquid separation steps are the same as in current usage. The clarified solution goes to ion exchange for uranium recovery and then on to tailings neutralization. The solid tails are treated in an added flotation circuit to remove pyrite concentrate. This concentrate is densified and fed to a new pressure leach circuit operating at about 160 C and 6 hours contact time. Sufficient air (or oxygen) must be used to convert about half the sulphur present to sulphate, and a 50% usage efficiency should be assumed. This heated wet air might be used to agitate the leaching pachucas as is currently done at the Rio Algom carbonate leach plant at Lisbon, just outside Moab, Utah. For a 1000t/d operation, the air rate would be about 2500 L/S or about 5300 c.f.m.

At the completion of the pressure leach reaction, the pressure leach discharge must be filtered and washed. A pressure filtration system is probably the preferred alternative to maintain the solution heat. A low soluble loss is not a necessity. The cake from the filtration system could be treated separately to maintain effective pyrite disposal.

This system has not been tested adequately to determine if it is a workable system. Nor has it been examined in detail from an engineering aspect. The advantages appear to be that: (i) it might reduce or eliminate sulphuric acid purchases for the leach circuit (ii) half or so of the pyrite is used up usefully (iii) the system seems simpler than a full scale acid plant (iv) a slight additional uranium extraction might be achieved from the pyrite.

To carry this suggestion further requires some more extended laboratory tests. These would include: (i) several pressure leach tests on a larger scale, filtering off the solution and using this hot solution for leaching fresh ore (the patent added the whole autoclave slurry to the fresh ore) (ii) settling/filtering tests on the pressure leach discharge and on the whole ore leach to assess solid/liquid separation (iii) ion exchange testing of the leach liquor to make sure the added iron (presumably ferrous iron) does not preclude ion exchange recovery. These tests should probably be performed at a laboratory with considerable pressure leach expertise. At completion of the bench scale tests, a further review of the practicality of the approach could be defined and larger scale or continuous tests run if warranted.

7.8 TAILINGS FLOTATION PLUS ACID PRODUCTION

This option is one which is used in South Africa, and has been studied in Elliot Lake. It would produce a much lower sulphide level in the tailings. However, acid in excess of that required for leaching Elliot Lake ore would be produced if all the mined ore was treated, and this acid would have to be marketed. The market for sulphuric acid is poor, although Elliot Lake has one advantage. It is close to the Great Lakes, and water shipping facilities.

If Elliot Lake became a net acid producer, it is conceivable that flotation plants could be built adjacent to idle tailings areas, which could then be treated by flotation to remove much of the pyrite. Again, this kind of recovery and treatment of tailings is being done in South Africa, but primarily because of the gold and uranium content of the pyritic material. Also, South Africa has few other sources of acid than this pyrite.

The option is certainly a "practical" way to reduce the pyrite level in the tailings at currently operating mines, and all the design requirements are available. It has not yet been seen as commercially viable. And the residual pyrite in the flotation tailings is still high enough to give environmental problems, without further treatment.

7.9 MEMBRANE PLUS COVER

While rigid covers are not considered practical approaches to pyrite oxidation control in uranium tailings, a flexible elastomer such as hypalon, combined with a protective "natural" cover could be used. The membrane would prevent oxygen from getting to the tailings, and water ingress would be restricted to the periphery and through any leaky seams or small holes in the liner. The liners which could be used, for instance hypalon or flexible PVC, have generally long life and good resistance to chemicals. They have been and are being used in many places in large water containment basins, and in impervious dams and as liners for settling ponds in Elliot Lake.

Membrane liners require careful laying to prevent any rupture or puncture during installation. The liner would have to be properly integrated into the sides of the tailings basin by concreting and grouting into rock abutments and careful burying in engineered structures. The liner will also require covering with sand, soil or some such material. The minimum cover depth would probably be in the 2 to 3 m range. Cover material next to the cover would have to be fine grained, with a defined maximum grain size. Structurally the material could probably be nearly any type of loose material in the area. It's major function is to protect the liner itself from damage. It does not have a structural stress requirement. The main requirements will be its capacity to absorb and discharge the precipitation of the maximum rainstorm expected. It should also be vegetated after placement so that the whole mass of tailings blends back into the surroundings. Depth of cover would depend on requirement to support trees, bushes, etc. with minimal damage because of roots. Properly laid and covered, the liner could be a long term solution, possibly a century.

The cost of such a liner-cover system would be substantial. On a 160 hectare (about 400 acre) tailings basin 1,600,000 m^2 (about 17 million square feet) of liner would be required. As well, some 4 x 10^6 m^3 of cover would have to be moved and almost 1 x 10^6 of this would have to be screened, and carefully laid as the first protective cover layer. The local supply of loose soil (not structural sand and gravel) would have to be assessed. At the completion of the program, the whole area would be revegetated.

The advantages of the method is that it provides a defined, long term program which should virtually prevent pyrite oxidation. The disadvantage is that it depends on a synthetic material which will probably eventually deteriorate badly enough to require replacement. Also there will certainly be some leaks which will allow air and water to get to the tailings. Probably the acid generation will be small enough so it would not be troublesome initially.

A test program to examine this alternate would have to be thoroughly reviewed by people knowledgeable both in soils and with such liners. It is unlikely that any meaningful data could be generated from an actual experiment. Oxidation rates of the pyrite would have to be inferred from air and water permeability tests on the chosen liner itself, and on the cover materials. A simulation run which would determine gross acid generation rates would then be needed, and its results used to determine the possibilities of the approach.

7.10 METHOD REVIEW

Of the nine approaches discussed in section 7, three refer to flotation. These three define differing ways of disposing of the flotation concentrate: viz - by burial, by pressure leaching with special disposal of the residue, or by roasting for acid manufacture. In all cases the tailings from the flotation circuit are still expected to contain enough sulphide to cause acid generation. Flotation does not stop acid generation, but because it substantially reduces the sulphide quantity present in the tailings at the surface of the basin it should make control easier.

The use of sodium lauryl sulphate can only be considered a short term expedient to get an existing basin which has turned acid under control - to reduce the oxidation rate substantially until some other control technique can be put into place.

Limestone can be considered as a potentially less expensive substitute for lime as it is currently used. It can also be considered as a long term delaying tactic which can reduce the rate of oxidation by maintaining a relatively high pH and preventing bacterial catalysis of pyrite oxidation. Combined with flotation, it seems possible the oxidation rate would be acceptably low for decades and perhaps centuries. But the potential pyrite oxidation possibility will still remain.

Apatite - phosphate rock - has the potential for control of acid generation and radionuclide emission. Insufficient study has been done on this approach. Potentially it may be more effective than limestone. It appears to be a method requiring considerably more research.

The three remaining methods are approaches for greatly reducing the oxygen which will come in contact with the pyrite. The synthetic membrane approach would be expensive. Some other impermeable barrier above the tailings and below a cover layer might be effective. Movement and screening of material to provide cover layer would be a large project, and the stripping of the soil from the vicinity of the tailings could cause substantial negative environmental impact. At coal mines in the U.S.A. where such approaches have been tried, the cover material appears to have been overburden which was moved initially.

Muskeg can only be effective in certain topographically suitable areas. It seems to rely primarily on maintenance of a water layer close to or at the tailings surface, and to fixation of oxygen at a layer above the water level. Peat bogs are a feature of Northern Ontario, so the approach is similar to Ms Kalin's - work with nature. Care must be taken to maintain the bog for the long term.

Deep water disposal appears to be a long term, maintenance free solution to reduce pyrite oxidation to negligible rates. For the "active" operations, it appears that most of the currently produced tailings could be treated by this method. The proposed technique does assume an ongoing active surface basin which will require some other treatment. The application of the method to an inactive tailings site has not been addressed. It would require reworking those inactive tailings, not considered a "practical" approach.

In summary, there appear to be potentially practical ways to virtually eliminate acid generation from pyritic uranium tailings. All will require long term observation and some sort of perpetual care. A cost comparison between methods has not been attempted.

8.0 CONCLUSIONS

1. The only appreciable quantities of pyritic uranium tailings in Canada are in the Elliot Lake area of Northern Ontario. The pyritic uranium tailings in Elliot Lake are only a part of the many sulphide containing deposits in Canada which are giving or will give acidic run-offs. The tailings in Elliot Lake now amount to about 8% of the tonnage of these Canadian sulphidic tailings, and cover about 3% of the area of such tailings.

2. The primary reason for substantial and rapid generation of acid and acid salts from such sulphidic tailings is the action of the bacterial species thio oxidaans. These microscopic creatures catalyze the oxidation phenomena which results in the production of sulphuric acid and acid salts.

3. The Elliot Lake tailings differ from the base metal tailings. In the case of base metals, the usual concentration method after grinding is flotation. In Elliot Lake, the ground ore is leached for a considerable time (36 hours) with strong (5 - 10%), hot (75 C) sulphuric acid. This leach removes any buffering minerals from the ore. Also, Elliot Lake tailings contain small amounts of radionuclides from both the thorium and uranium natural radioactivity decay series.

4. On abandonment, the impact of acid releases into streams and rivers from untreated pyritic uranium tailings would spread far beyond the tailings basins themselves. The expected amount of these acidic releases can be estimated from studying the existing inactive tailings basins. The effects on natural streams can be estimated by considering present stream flow patterns.

5. It seems likely that the pyrite in the tailings below the lowest level reached by the water table, in an abandoned and untreated tailings area, will not be oxidized at any significant rate by bacterially catalyzed oxidation phenomena. It also seems likely that all the pyrite in the tailings deposited above the lowest level of the water table will, in the long run, be oxidized. The oxidation rate will depend on the size consist, the permeability of the tailings to oxygen, the concentration of sulphide sulphur present, the depth of the sample, and the water availability. While research is being done in this field, so far no empirical equations combine all the pertinent factors.

When surface zone oxidation occurs, all of the interstitial water within the whole tailings basin will become acidic because of the downward migration of acid solutions over time. Pyrite can be oxidized and solubilized in the zone below the water table by ferric iron entering in the acid solution from the top zone. This chemical oxidation would significantly increase the overall amount of pyrite which will be oxidized.

Maintaining the water table as high as possible, and minimizing the pyrite in the upper layers of the tailings will minimize the amount of pyrite oxidized.

6. Deep water disposal appears to be the best disposal method for an operating mine producing pyritic uranium tailings. If a combination of a surface site and a deep water site are used, then the impact from the tailings which are deposited underwater will be low, and no stream diversions or extra treatment will be necessary. So long as the lake remains, these tailings should never generate acidity at an appreciable rate.

7. Even if deep water disposal cannot be used, converting tailings areas to bogs, and providing coverage of organic material and bog plants should completely stop pyrite oxidation. Such natural systems will only be possible in limited tailings configurations, but where they can be used they should be effective. The muskeg material of a bog provides a living oxygen absorption blanket. Immediately below this blanket, water is necessary to maintain the cover alive. Under these conditions, vegetation will prevent pyrite oxidation; indeed, anaerobic bacteria beneath such a cover could convert sulphates to sulphides.

8. Vegetation can only be considered a way of stopping pyrite oxidation under special circumstances such as the marsh conditions mentioned above. Vegetation on elevated tailings areas, as presently practised in the Elliot Lake area, has many advantages, but it cannot be considered a cure for pyrite oxidation.

9. Rigid cappings at surface, such as cement or asphalt, would require extensive foundation and drainage preparation, are not aesthetically desirable, and would crack under frost heaving and ground movement. Such cappings do not appear to be practical approaches.

Studies on acid sulphate soils indicate that covers of loose materials, such as sand or gravel or soils, might slow the oxidation of pyrite, but not sufficiently to remove the need for continued treatment, unless the increase in the water table occasioned by the cover completely immersed the pyritic material.

A flexible impervious layer immediately above the pyritic tailings, with a top cover of some protective material should prevent pyrite oxidation as long as the flexible layer remains impervious to oxygen. Such an impervious layer could be provided by a synthetic membrane or a compacted clay-till mixture. The difficulty of this approach includes the unknown life of synthetic membranes, and the absence of appreciable clay in the Elliot Lake area. As well, the quantity of cover material is substantial. In the case of a synthetic membrane, the cover directly above the membrane would have to be of screened and selected material.

10. Stabilization of the tailings by additives which produce rock-like structures is unlikely to be a good solution. No data is available or could quickly be made available on the length of time these stabilized structures would continue to exist. Such methods may be advantageous in some particular cases, but overall they cannot presently be considered practical alternatives.

11. Pyrite flotation is not a panacea for stopping pyrite oxidation, but it does substantially reduce the potential for oxidation. In conjunction with calcite additions to the depyritized tailings, and proper disposal of the pyrite concentrate, several decades will probably elapse before tailings treated in this manner will generate substantial quantities of acid.

12. None of the process innovations being examined by research laboratories specifically address the pyrite oxidation potential of Elliot Lake tailings. The products of pyrite oxidation will be the major environmental impact from these tailings. Process development studies should address this problem.

13. While institutional control over the long term has been considered, climatic variations should also be considered. The long term warming trend which seems to be accepted because of CO_2 build up in the atmosphere may influence tailings disposal methods. The use of bog cover, and maintaining a high water table level might not be applicable if a marked warming and drying trend influenced Northern Ontario weather in the next century.

14. Other groups in Canada and the U. S. are addressing the problems of sulphide oxidation. Information exchange between these groups and National Uranium Tailings Program personnel will help develop a reasonable close down method for pyritic uranium tailings.

9.0 RECOMMENDATIONS

1. As much lead time as possible should be given the mining companies ahead of actual close down. Procedures which modify the top several feet of tailings can be done relatively cheaply and efficiently during the operating period; they effectively become non-options when all the tailings have been laid down. Such procedures include flotation, and intimate blending of materials such as limestone or apatite in the top few feet of the tailings layers. A defined close out procedure should be agreed upon between the mines and the regulatory bodies at least five years before actual shutdown.

2. Process research should address the major chemical problem in the Elliot Lake tailings: pyrite oxidation. All the mining operations are now in place, and any major process change could not be reasonably implemented. The objective of process research should be to devise a sulphuric based system which will minimize pyrite oxidation. A hypothetical example of such a system would be pyrite flotation of the feed to leach, and treatment of the uranium containing pyrite concentrate and the uranium containing tailings in two separate leach circuits. The conventional sulphuric acid leach would be used on the pyrite concentrate. The residue from concentrate leaching would be treated so that it could never generate acidic solutions which would enter the environment (pumped underground, deep burial, etc.)

The leach on the low pyrite material would add an oxidant to convert all the remaining pyritic sulphide to sulphate. The tailings from this leach would, then, never be able to generate acid.

3. The active tailings basins should be examined with a view to assessing how the water table could be kept as high as possible naturally. This will minimize the volume of tailings which can turn acidic under the catalytic action of bacteria.

4. Tests should be run on the use of sodium lauryl sulphate as a method of substantially reducing the rate of pyrite oxidation in old tailings. Tests should also be run on encapsulated SLS, and on the newer bactericides such as the sorbates and the benzoates. These have the potential for longer term (several years) inhibition of bacterial action.

5. The potential use of living organic covers such as muskeg should be tested on a large scale under conditions which would allow the tailings to act as bogs. Those tailings in Elliot Lake which are susceptible to this sort of treatment should be examined to see if the method might be applicable.

6. Deep water disposal should be considered in conjunction with the current land based systems to determine if the combined route may not lead to close down conditions with lower potential for generation of pollutants, at lowered overall costs.

7. A test program should determine if the addition of ground apatite to the tailings will reduce acid generation rates. The other effects of phosphate, for instance, eutrophication, should be considered in any final evaluation.

8. The role of limestone should be assessed as a potentially cheaper method of solution neutralization than lime, particularly after close down. This would require a cost study of the two reagents as the position would be after close down. This study might help develop co-operative methods of handling close down between the two mining companies in the Elliot Lake area.

9. The role of limestone in keeping the oxidation of pyrite under control by preventing the development of the bacterial oxidation should be further reviewed. Large scale tests to explore this option, and to determine how to replace limestone periodically should be considered.

10. The permeability of large expanses of a synthetic membrane to water and oxygen should be established. The physical problems of installing an impervious blanket at the surface of the tailings, and covering it with a non-pyritic layer should be considered. The problems might include the effect of the impervious membrane on the tailings structures, and on the interstitial water which would lie between the vegetative growth at surface and the impervious layer. The effect of prolonged drought should be considered. The costs of this option, and the availability of the necessary cover material at Elliot Lake sites should be critically assessed.

11. Tests should compare the natural oxidation rate of tailings with the rates on flotation tailings and on flotation concentrate. These tests would determine the effect of sulphur content on rates of oxidation.

12. Every effort should be made to keep up to date with research and development on sulphide oxidation and its control. This should include reviewing developments in the U. S. and other countries such as South Africa, Australia, and those in Western Europe, as well as in Canada.

10.0 ACKNOWLEDGMENTS

Many people provided information which is included in this report. Many not specifically included below have provided insights.

Dr. K. C. Ivarson of the Canadian Department of Agriculture discussed the formation of acid sulphate soils, and provided several pertinent references.

Mr. Ron Nicholson of the University of Waterloo reviewed the use of carbonates in keeping the oxidation rate of pyrite under control, and discussed his experiments. He has contributed calculations regarding the rate mechanism in the text.

Dr. Robert Kleinmann of the U. S. Bureau of Mines sent literature on the use of sodium lauryl sulphate, and on the mechanism of acid generation from pyrite oxidation, and discussed this work briefly on the telephone.

Dr. Alfred Stiller of the University of West Virginia described his work on phosphate rock as a way of stopping pyrite oxidation in pyritic coal wastes, and sent a package of literature on acid mine drainage research.

Mr. Phinney, of the Monenco group, which is working on a report on oxidation of base metal sulphides, sent a list of references. Mr. Ferguson of INCO kindly let me review the first draft of the Monenco report.

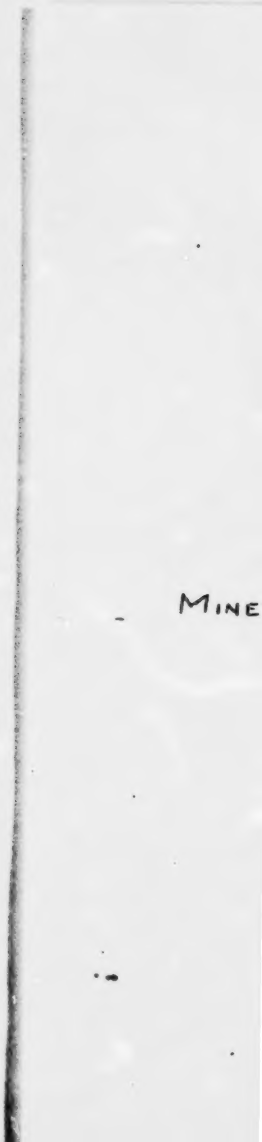
Ms. Margaret Kalin of the University of Toronto School of Environmental Studies spoke of her work on tailings, and provided a description of her proposed approach.

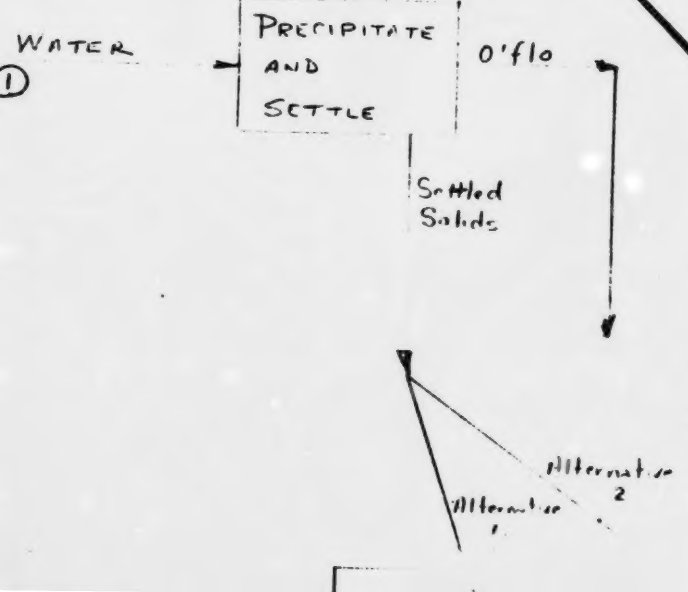
Dr. Chakravatti of Denison discussed some parts of the draft report. Both Dr. Chakravatti and Mr. Rickaby provided information on Denison operations.

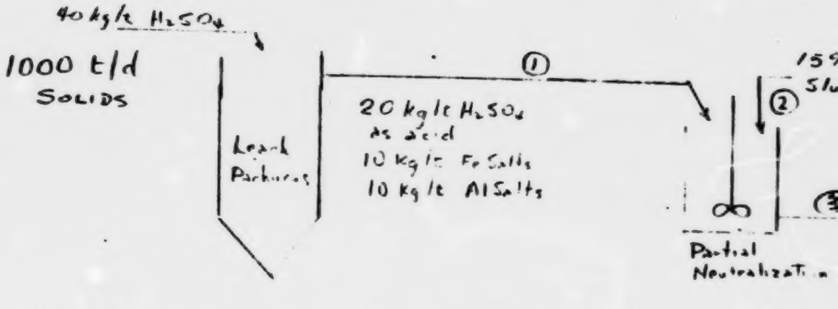
Mr. John Hawley of the Ministry of Environment of Ontario provided MOE documentation of early tailings problems in Elliot Lake. Mr. G. Lahaye provided further help and documentation.

Dr. Nan Dave and Mr. T. P. Lim of the CANMET laboratories in Elliot Lake discussed their work on the Nordic tailings, and their experiments on underwater disposal. Messrs Gow, Campbell, Ritcey and Lucas of CANMET in Ottawa provided helpful comments. Dr. M. Silver discussed the lysimeter test program at CANMET, and Dr. J. Skeaf commented on chlorine testing and CCIW lysimeters.

Rio Algom personnel, including Messrs Kay, Barnes, Roch and Culver, but particularly Mr. Vivyurka, have been extremely helpful. Rio Algom supplied virtually all the background information on present run-off and control of effluents from inactive tailings areas. Rio Algom are to be complimented for setting up and maintaining such a fine system for information gathering and data collation, and for making the information readily available to those studying the problem. The kind of background they are supplying will be of immense help in developing better ways of ameliorating the unwanted effects of pyrite oxidation in uranium tailings.


MINE

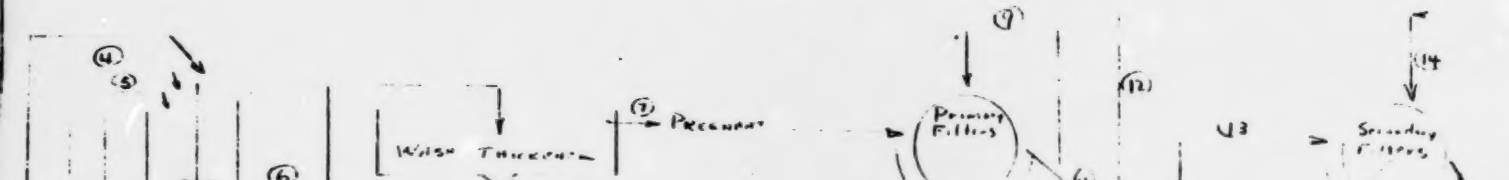

WATER
①
PRECIPITATE AND SETTLE
O'flo
Settled Solids
Alternative 2
Alternative 1
H_2SO_4


40 kg/t H_2SO_4
1000 t/d SOLIDS
Leach Pachucas
①
20 kg/t H_2SO_4 as acid
10 kg/t Fe Salts
10 kg/t Al Salts
15%
Slu
②
Partial Neutralization

STREAM	①	②	③	
DESCRIPTION	Leach Discharge	Lime Slurry	Partial Neut Discharge	Se F t.
% SOLIDS	70%	15%	67%	
TONNES LIQUID/DAY	429	70 (Slurry) 4 (Reaction) 74 Total	503	5
Kg U_3O_8	1000	0	1000	
Estimated pCi/L Ra	—	0	—	

EXPANDED SOLID

(Note. Figur


(4)
(5)
#1 Repulper
(6)
Wash Thickener
(7) Pregnant
(8)
Primary Filtrate
Secondary Filtrate
(5)
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(9)
Primary Filters
(11)
(12)
#2 Repulper
(10)
(13)
Secondary Filters
(14)
(15)
#3 Rep

(5) Primary Filtrate	(6) Wash Thickener Feed	(7) Pregnant to Ion Exchange	(8) Wash Thickener Underflow	(9) Secondary Filtrate as Wash	(10) Total Secondary Filtrate	(11) Primary Filter Cake	(12) #2 Repulper Dilution	(13) Secondary Filter Feed	(14) Secondary Filter Wash	(15) Secondary Filter Cake	T
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D SEPARATION AND SOLIDS WASHING CIRCUIT WITH MASS BALANCE

kg U3O8's in streams are not completely rationalized- some inaccuracies)

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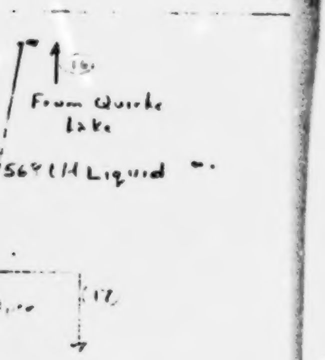
16
From Quirke Lake
56% CH Liquid
17

16	17 Tertiary [illegible] Discharge
0	52%
511	818
0	4
3	4

ALTERNATIVE

2

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APPENDIX A

STATEMENT OF WORK

FROM NUTP CONTRACT

14SQ.23241-3-1668
OSQ83-00278

APPENDIX A

PRACTICAL CONSIDERATIONS OF PYRITE OXIDATION CONTROL IN URANIUM TAILINGS

STATEMENT OF WORK

Background:

The natural oxidation of sulphides contained in mine wastes is recognized as an environmental concern. Most of the mine/mill wastes produced in Canada contain significant concentrations of sulphides of iron and heavy metals that produce sulphuric acid and thiosalts that can acidify the surrounding water shed and promote dissolution of metals that exhibit various degrees of toxicity. There are currently over 100 million tonnes of uranium tailings in Canada that contain significant quantities of acid-producing sulphides and this is being augmented at the rate of 7 million tonnes annually. The quantity is expected to be double present quantity by the year 2000.

This quantity, although small in comparison to total Canadian mine tailings, is significant because natural acidification in uranium tailings facilitates mobility of radionuclides, particularly isotopes of thorium and radium. Over 99% of the acid-producing uranium tailings are located in the Elliot Lake-Blind River areas of Ontario. The mechanism of acidification has been, and continues to be subject to research by several Canadian agencies and universities and several possible solutions to the acidification problem have been suggested. However, no practical solution to the problem has yet been applied. Drainage from inactive sites continues to be treated, a practice that is neither acceptable to uranium mining companies nor to the public through its agencies, in the long-term.

Statement of Work:

An examination of alternatives to alleviate the problem of acidification of uranium mine tailings is required. This examination will include a review for practical implications of mitigation measures that have been suggested from field observations and laboratory testing. The details of past practice to control pyrite oxidation and the results to-date will be determined. For possible mitigation measures, test programs will be proposed that will predict the long-term success of these measures. The contractor will undertake the following work:

1. Define the problem of pyrite oxidation in the uranium mill tailings in the Elliot Lake - Blind River areas of Ontario. The definition will include the determination of pyrite content in the various mill tailings, the total tonnage and the known extent of oxidation since the commencement of ore processing in 1955. Determine the contaminants of concern and the effects on the water shed at the representative locations.

.../2

2. Describe the development of practices at the Elliot Lake - Blind River uranium mines to control the oxidation of pyrite and to treat acidic drainage, and the effect these practices have shown.

3. Describe practical ways to modify existing operating tailings and storage systems in Elliot Lake to mitigate pyrite oxidation in the tailings after operations cease. Propose test work which is required to confirm design and implementation of these modifications.

4. Describe practical ways to modify inactive tailings storage systems to prevent pyrite oxidation upon abandonment. Estimate economic and environmental consequences of these modifications. Describe test work which is required to confirm design and implementation of these modifications in the long-term. For purposes of this study, the long-term is in excess of 100 years.

5. Determine the applicability of Canadian research findings on the control of pyrite oxidation in uranium tailings in the Elliot-Lake - Blind River areas.

APPENDIX B

LIST OF REFERENCES

REFERENCES

1. "A Review of International Uranium Mill Tailings Management Practice" by J.B. Davis, V. Milligan, and J.L. Seychuk - paper 23 - presented at Uranium '82 - 12th Annual Hydrometallurgical Meeting, Toronto, Canada -Aug 29 - Sept 1, 1982.

2. "Mobility in Ground Water of Contaminants from Uranium Mill Tailings" by J.A. Cherry - paper 24 - presented at Uranium '82 (see above).

3. "Tailings Disposal by the Thickened Discharge Method for Improved Economy and Environmental Control" by Eli J. Robinsky, Assoc. Prof. of Engineering, University of Toronto and E.J. Robinsky Associates Ltd., Toronto,Canada.

   An informal paper given at the "short course" (below).

4. "Tailings Treatment and Solution Recycle" by D.M. Gorber, B.G. Ibbotson, & R.A. Knapp - Lecture 13 - Extractive Metallurgy of Uranium - A Short Course - University of Toronto - May 1978.

5. "Environmental Aspects of Uranium Mining & Milling Operations" by D.M. Gorber, B.G. Ibbotson, & R.A. Knapp - Lecture 14, U of T short course (above).

6. "The Physical, Chemical & Radiological Characteristics of the Surface Layers of Ontario Uranium Tailings Deposits - 1982" by J.R. Hawley - Head, Mining, Metallurgy & Inorganic Chemistry Unit, Quality Protection Section, Water Resources Branch, Ministry of the Environment of Ontario.

   Two tables from this report showing a range of values for Elliot Lake tailings surface layers are included in Appendix 1.

7. "Serpent River Basin Water Quality Data - 1980". Issued by C.E. McIntyre, Director, Northeastern Region, Ministry of Environment of Ontario - September 1981.

8. "A Report on an Industrial Waste Survey of Denison Mines Limited" - August and September 1968 by Division of Industrial Wastes - Ontario Water Resources Commission. Report by John Hawley.

9. "Quirke Lake Tailings Disposal Study - Stage 1 - Background Information" - Rio Algom Limited, Denison Mines Limited with Senes Consultants Limited & Golder Associates - October 1981.

   This work provides background by examining underwater disposal as used elsewhere, a physical study of the Quirke Lake basin, a study of placement methods, a study on the effect on underground workings, and a study on development of a model of tailings disposal.

10. "Mine Tailings Disposal Options - Social and Economic Impact Analysis" - Elliot Lake, Ontario - April 1983 - by ESA in conjunction with MIE Ltd., Consulting Engineers.

    Reviews the work above, and produces cost analyses which indicate underwater disposal may be the cheapest disposal alternative. The back-up data on which cost analysis is based were not provided.

11. "Microbiological Transformations of Iron and Sulfur and Their Application to Acid Sulfate Soils and Tidal Marshes" by K.C. Ivarson, G.J Ross & N.M. Miles. Chap. 4 of "Acid Sulfate Weathering" published 1982 by Soil Science Society of America, 677 South Segoe Road, Madison, Wisconsin, 53711. Edited by D.M. Krol. Information from a symposium at Fort Collins, Colorado, Aug. 5-10, 1979.

12. "Microbial Formation of Basic Ferric Sulfates in Laboratory Systems and in Soils" by G.J. Ross, K.C. Ivarson, and N.M. Miles. Chap. 5 from "Acid Sulfate Weathering" (above).

    The formation of jarosites by thiobocillus in acid sulfate soils is examined in laboratory tests. Field studies of Canadian soils is also described.

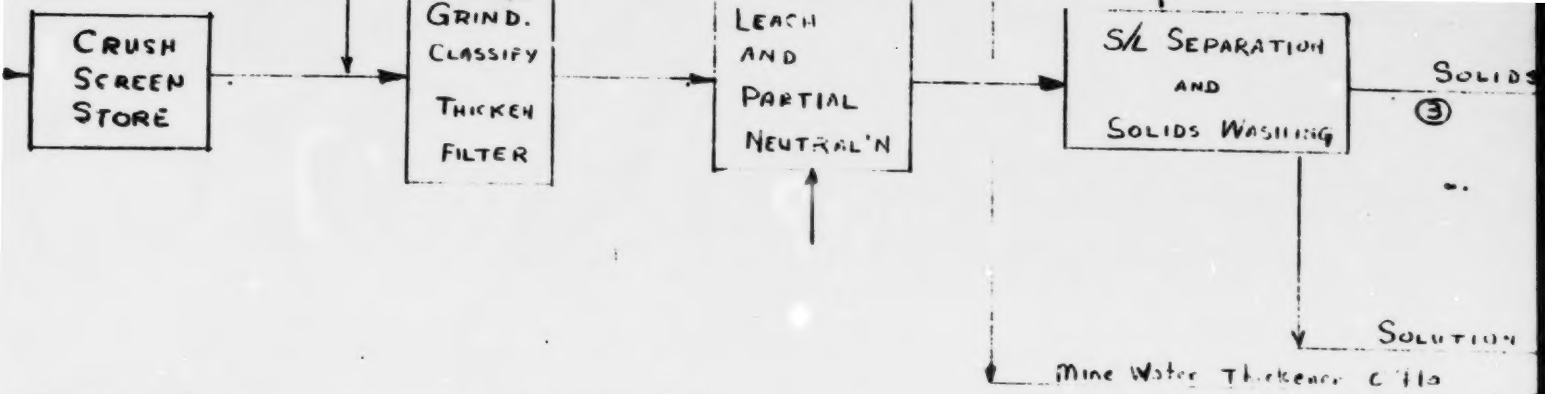
13. "Characteristics and Reclamation of "Acid Sulfate Mine Spoils" by R.J. Bornhisel, J.L. Powell, G.W. Akin, and M.W. Ebelhar. Chap. 13 from "Acid Sulfate Weathering" (above).

    Notes that many factors limit establishment of vegetative cover on mine spoils. Also refers to legislation in U.S. on rehabilitation.

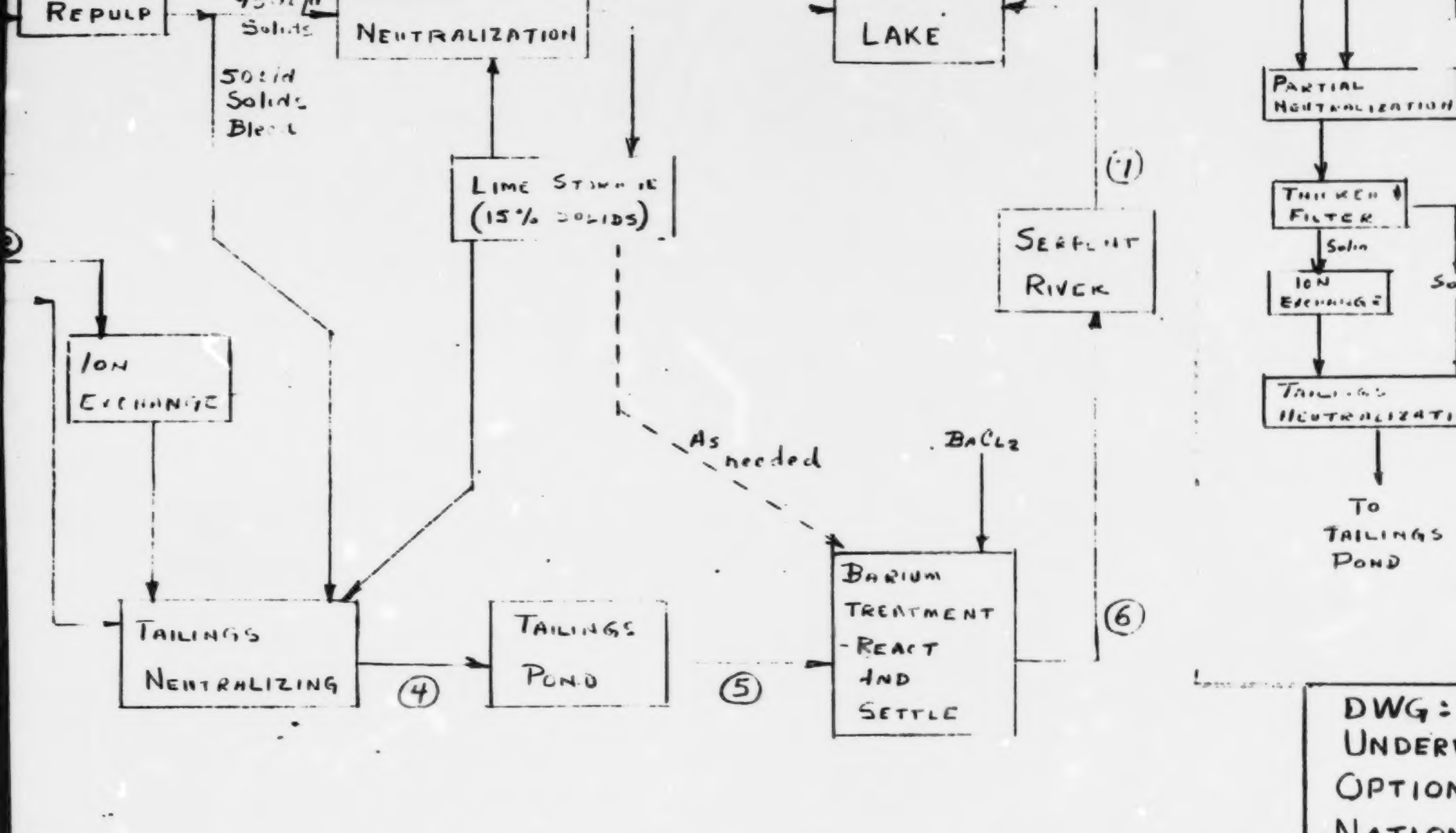
14. " An Annotated Bibliography of Surface - Mined Area Reclamation Research" - October 1983 - from U.S. Dept. of Agriculture, Forest Service, Northeastern Forest Experiment Station, Route 2, Kentucky Highway 21 East, Berea, Kentucky 40403.

    Covers all publications from Forest Service since 1962 - 172 references. Covers Overburden & spoil properties, Streamflow & water quality, Revegetation for wild life & reforestation, and Landscape quality.

MINED ORE


CRUSH
SCREEN
STORE
GRIND.
CLASSIFY
THICKEN
FILTER
LEACH
AND
PARTIAL
NEUTRAL'N
S/L SEPARATION
AND
SOLIDS WASHING
SOLIDS
③
SOLUTION
Mine Water Thickener C'flo

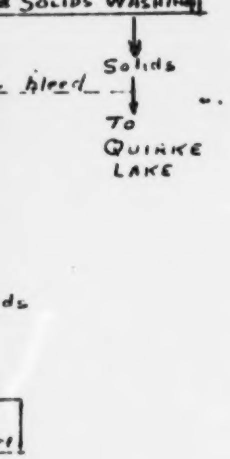
STREAM NUMBER	①	②	③	④	⑤	⑥	⑦	⑧
STREAM NAME	MINE WATER	PREGNANT LIQUOR	FILTER DISCHARGE	TAILS NEUT DISCHARGE	TAILS POND EFFLUENT	SET'L'G POND EFFLUENT	SERPENT RIVER	QUIRKE LAKE
m^3/d LIQUID	460	1196	250	—	—	—	—	1820
pH	2.8	2.2	~ 4	~ 9	4.2	8.2	7.2	6.3
——	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
SOL. FE	97	1600	—	1.5	30	1	0.2	—
ACIDITY	601	—	~ 6	–	~ 120	—	6	4
NH_3/N	48	273	~ 6	78	54	49	13	4
NO_3/N	69	1260	~ 10	575	62	49	19	8.3
TDS	1400	22,000	~ 350	3,000	2500	2400	870	320
—	pCi/L	pCi/L	pCi/L	pCi/L	pCi/L	pCi/L	pCi/L	pCi/L
RADIUM	190	~ 600	~ 5	~ 1500	~ 300	15	7	3


REPULP
Solids
NEUTRALIZATION
LAKE
50% Solids Bleed
LIME STORAGE (15% SOLIDS)
(7)
SERPENT RIVER
PARTIAL NEUTRALIZATION
THICKEN FILTER
Solid
ION EXCHANGE
TAILINGS NEUTRALIZATION
To TAILINGS POND
ION EXCHANGE
As needed
BaCl2
BARIUM TREATMENT - REACT AND SETTLE
TAILINGS NEUTRALIZING
(4)
TAILINGS POND
(5)
(6)

BLOCK FLOWSHEET & ESTIMATED STREAM COMPOSITION

DWG:
UNDER
OPTION
NATION
TAILIN
DSS-05Q83-
K.D.HES
March 29/84

Revised Apr 16/84
KDH


SOLIDS WASHING
Solids
bleed
TO QUIRKE LAKE
ds

NUTP-H-01-Rv1
WATER DISPOSAL
- 1000 TONNES/D
AL URANIUM
S PROGRAM
0278/14SQ-22341-3-1668
TER & ASSOCIATES

15. "Contaminant Migration in Seepage from Uranium Mill Tailings Impoundments - An Overview" by T.A. Shepherd & J.A. Cherry. Symposium of Uranium Mill Tailings Management, Fort Collins, Colorado - Nov. 24-25, 1980.

    Authors discuss neutralization path length, neutralization capacity, & neutralization ratio in examining the hydrology and the hydrochemistry of tailings deposits. They define advection & dispersion. Excellent review of uranium tailings impacts.

16. "Groundwater Monitoring and Contaminant Occurence at an Abandoned Tailings Area, Elliot Lake, Ontario" by R.D. Blair, J.A. Cherry, T.P. Lim, & A.J. Vivyurka. First International Conference on Uranium Mine Waste Disposal, Vancouver, B.C. - May 19-21, 1980 - S.M.E. of A.I.M.E. Proceedings Volume - C. Brawner, Editor.

    Reviews early history of Nordic and discusses studies done on the Nordic tailings to examine geological pattern below the tailings, existing aquifers, and production of acid within the tailings themselves.

17. "Subsurface Hydrology and Geochemical Evolution of Inactive Pyritic Tailings in the Elliot Lake Uranium District, Canada" - by J.A. Cherry, R.J. Blackport, N. Dubrovsky, R.W. Gillham, T.P.Lim, D. Murray, E.J. Reardon, & D.J.A. Smyth. Symposium on Uranium Mill Tailings Management, Fort Collins, Colorado - Nov. 24-25, 1980. (Geo Technical Engineering Program, Civil Engineering Dept., Colorado State University.

18. "Control of Acid Drainage from Coal Refuse using Anionic Surfactants" by R.L.P. Kleinmann, and P.M. Erickson. U.S. Bureau of Mines, R.I. 88747, 1983.

    Reviews pyrite oxidation, discusses the use of sodium lauryl sulphate (SLS) as an inhibitor, describes spray application of SLS and describes tests to determine requirements of the chemical.

19. "Acid Mine Drainage: Control and Abatement Research" by A.G. Kim, B.S. Heisey, R.L.P. Kleinmann & M. Deul. U.S. Bureau of Mines. I C 8905 (1982).

    Reviews the acid mine drainage problem in coal mines, quantifies the problem, and discusses control strategies.

20. "Biogeochemistry of Acid Mine Drainage and a Method to Control Acid Formation" by R.L.P. Kleinmann, D.A. Crerar, & R.R. Pacelli. Mining Engineering - March, 1981.

    Considers the mechanism by which the thio bacillus works and defines some of the rate controlling mechanism. Discusses field tests using SLS and the theoretical reason this works.

21. "The Recovery of Elemental Sulphur from Pyrite and Pyrrhotite" by K.W. Downes & R.W. Bruce - C.I.M. Bulletin - March, 1955 - Trans. Vol. LVIII pp 77-82.

    Describes how an artifical pyrrhotite can be made from pyrite generating sulphur, and how this pyrrhotite can be converted to elemental sulphur.

22. Canadian Patent Specification 722,143 "Extraction of Elemental Sulphur from Sulphides of Iron".

    This patent specification assigned to the Minister of Mines and Technical Surveys applied for on July 31, 1951 and published Jan. 19, 1955. The patent was contested. It describes the recovery of elemental sulphur from artifical pyrrhotite, made by heating pyrite to eliminate the labile sulphur.

23. U.S. Patent 3,273,972 "Sulphuric Acid Leaching of Uranium Bearing Material" by R.A. Campbell, J.D. Elliott, J.R. Fisher & V. Opratko, assigned to Rio Algom Mines - Sept. 20, 1966.

    Describes the production of a "leaching slurry" by flotation of sulphides from feed or tailings, or from foreign pyrite.

24. "Treatment of Pyrite by Low Temperature Thermal Decomposition and Aqueous Oxidation" - by W. Kunda, B. Rudyk & V.N. MacKew. A paper presented at the Conference of Metallurgists, Kingston, Ontario - Aug. 28-30, 1967.

    Describes a process for thermally (pyrometallurgy) springing S from F_eS_2 and pressure leaching the resulting artifical pyrrhotite to produce elemental sulphur.

25. "Economics of Iron & Elemental Sulphur Recovery from Pyrite using Low Temperature Thermal Decomposition and Aqueous Oxidation" by B.R. Mehta & P.T. O'Kane.

    Provides costs for process described in the preceeding reference.

26. "Continuation Study on Hydrogeochemical Processes in Abandoned Uranium Tailings from the Elliot Lake District by J.A. Cherry, R.W. Gillham, E.J. Reardon, N.M. Dubrovsky, & R.V Nicholson - May 1, 1982 to March 31, 1983.

    Covers laboratory studies on pyrite oxidation kinetics at high pH levels.

27. "Pilot Study on Pyrite and Radio Isotope Removal by Flotation" by T. Eric Pakkala, Denison Mines Ltd. Nov. 23, 1978.

    Describes a large scale plant test on flotation.

28. "Lysimeter Investigations on Uranium Tailings at CANMET" by G.M. Ritcey & M. Silver - C.I.M. Bulletin 75 No 846 October 1982.

    Results of accelerated tests each on 500 lbs of tailings.

29. "Hydrochloric Acid Leaching of an Elliot Lake Uranium Ore" by K.E. Hague, B.H. Lucas & G.M. Ritcey - October 1979. Report ERP/MSL 79-140J. (for publication in C.I.M. bulletin).

    Discusses hydrochloric acid and chlorine assisted leaching of Elliot Lake ores.

30. "Chlorine Assisted Leaching of Typical Canadian Uranium Ores" by K.E. Hague. Report ERP/MSL 80-86 OP. (For presentation at the C.I.M. Conference of Metallurgists - Halifax, N.S. 1980).

    Discusses two stage HCl leaching of various Canadian ores.

31. "Aqueous Chlorine Leaching of Typical Canadian Ores" Dec. 1980 report ERP/MSL 80-167 (J). Published in "Uranium" by Elsevier (1982).

32. "Survey of the Occurence of Ra-226 in the Rio Algom Quirke I Uranium Mill, Elliot Lake" by J.M. Skeaf. C.I.M. Bulletin 74 No 830, June 1981.

33. "Preconcentration of a Low Grade Uranium Ore in C P D U and Laboratory Investigations to Optimize the Dewatering Conditions of the Preconcentration Products" by M.A. Cristovic et al, April 1982, CANMET Project MRP 3.5.0.0.01 "Control of Uranium Tailings".

34. "Migration of Contaminants from Underwater Buried Uranium Tailings - Laboratory Investigations of Radium, Total Dissolved Solids, Calcium and Sulphate Transfer Flux" by N. K. Dave & T. P. Lim - CANMET - Elliot Lake Laboratory, Nov. 1983.
    MRP/MRL 83 - 105 (OP) J. (To be presented at the 6th Annual Symposium on Management of Uranium Mill Tailings, Ft. Collins, Colorado, Feb 1 - 3, 1984, and to be published in the Proceedings.)

35. "Final Report: Immobilization of U - Th - Ra in Mine Wastes" by J.R. Brown, W.S. Fyfe, F. Murray & B.I. Kronberg - University of Western Ontario. Canadian Mining Journal, March 1982.

36. "Joint Panel on Occupational & Environmental Research for Uranium Production" - 1982 Annual Report - Published by Mining Research Laboratories - CANMET. Secretary Dr. R.O. Tervo, Manager, Elliot Lake Laboratory, CANMET.

37. "An Environmental Study of the Weathering of Mill Tailings in Connection with Water Pollution - Part 4 - The Effect of Tailings Samples from Various Canadian Mining Companies on the Composition of the Water" by A. Jongejan, Mines Branch, Mineral Sciences Division, IR 73-55. Sept. 18, 1973.

38. "Modelling of the Underwater Disposal of Uranium Mine Tailings in Elliot Lake" by B.E. Halbert, J.M. Scharer, J.L. Chakravatti, & E. Barnes. International Symposium on Management of Wastes from Uranium Mining & Milling - International Atomic Energy Agency, OECD Nuclear Energy Agency, Albuquerque, New Mexico, U.S.A. - 10-14 May 1982. Paper IAEA-SM-262-4.

39. "Hydrogeological Investigations and Evaluation of the Stanleigh Mine Tailings Impoundment Site" by J.M. Boyd, T. G. Carter, R.A. Knapp & K.B. Culver. Paper IAEA-SM-262-5 (from Conference above).

40. "Close Out Concepts for the Elliot Lake Uranium Mining Operations" by K.B. Culver, J.L. Chakravatti, D.M. Gorber, R.A. Knapp & J.B. Davis. Paper IAEA-SM-262-7 (from Conference above).

41. "Water Pollution from the Uranium Mining Industry in the Elliot Lake & Bancroft Areas". Vol. 1 - Summary, by J.R. Hawley - OWRC.

42. "Industrial Wastes Survey of Stanrock Uranium Mines Ltd. Elliot Lake, Ontario" 1969 - by J.R. Hawley - OWRC.

43. "An Industrial Wastes Survey of Rio Algom Mines Ltd. - Buckles, Lacnor, Nordic, Elliot Lake Ontario" 1970, by J.R. Hawley, OWRC.

44. "The Problem of Acid Mine Drainage in the Province of Ontario" by J. R. Hawley - 1972 - Ministry of Environment of Ontario.

45. "Environmental Assessment of the Proposed Elliot Lake Uranium Mines Expansion" Vol. 1, Background Information - March 1977. by James F. MacLaren, Ltd.

6. "Quantitative Determination of the Acid Producing Potential of Toxic Overburden" by J. J. Renton, A. H. Stiller, & T. Rymer. Obtained from Dr. Stiller.

47. "Modification of Acid Mine Drainage in a Freshwater Wetland" By R. K. Wieder, G. E. Lang & A. E. Whitehouse Symposium on Wetlands of the Unglaciated Appalachian Region, West Virginia Press, Morgantown, May, 1982.

48. "Research into the Use of Apatite Rock for Acidic Drainage Prevention" By F. A. Week.From Dr. Stiller.

49. "Induced Alkaline Recharge Zones to Mitigate Active Acidic Seeps" by F. T. Caruccio, G. Geidel, Obtained from Dr. Stiller.

    Discusses uses of alkaline recharge zones where the acid mine water is neutralized by limestone.

50. "A Method for Prevention of Acid Mine Drainage" by A. H. Stiller, Chemical Engineering Dep't, West Virginia University, Morgantown, West Virginia, 26506, USA.

    Discusses use of $Ca_5(OH)(PO_4)_3$ to tie up Fe^{+++}

51. "A Method for Prevention of Acid Mine Drainage: An Update Progress Report" By A. H. Stiller, May 3, 1983.

52. "At Source Control of Acid Mine Drainage: An Update" by K. J. Ladwig, P. M. Erickson, R. L. P. Kleinmann, & S. J. Onysko. U. S. Bureau of Mines, Pittsburgh Research Center, Pittsburgh, PA 15235.

    Discusses use of benzoate and sorbate materials as bactericides for control of Acid Mine Drainage.

53. "Water, Friend or Foe in the Control of Acid Mine Drainage" by K. L. Dyer, Northeastern Forest Exp. Stn. Rt. 2, Hwy. 21 East, Berea, KY 40403.

54. "Evaluation of Bentonite for the Control of Acid Drainage from Surface Mined Lands" by J. T. Crews & K. L. Dyer Northeastern Forest Exp. Stn. (see 53)

55. "report on Radiological Water Pollution in the Elliot Lake and Bancroft Areas" by Deputy Ministers' Committee. Issued November 24, 1965.

Committee appointed by John Robarts Nov. 1964. Excellent review of early problems in Elliot Lake with a chronology of start-up difficulties.

56. "Interim Report on Radiological Water Pollution in the Elliot Lake and Bancroft Areas - 1966" by the Ontario Water Resources Commission.

57. "Water Pollution from the Uranium Mining Industry in the Elliot Lake and Bancroft Areas". Vol. 2 - Detailed Findings. by J. R. Hawley OWRC March, 1972

Background information for the report in Ref 41

58. "Some Implications of Autotrophic Sulphur and Iron in Water Pollution by Uranium Mill Tailings in the Elliot Lake Area" by Frank Thompson - bacteriologist. OWRC November 1969.

A good review of the thio bacillus effects.

59. "Waste Disposal from Uranium Operations in the Algoma Watershed" by K. D. Hester & G. R. Yourt. 7th Annual Waste Conference, OWRC, June 6, 1960.

60. "Compost Production from Municipal Wastes of Canadian Mining Towns" by A. Jongejan CANMET. Presented By G. M. Ritcey at CUPMC, Annual Meeting May 19 - 20 1982, in Bancroft, and published in Proceedings.

61. "Problems in the Separation of Radium from Uranium Ore Tailings" by F. G. Seeley Hydrometallurgy 2 (1976/1977) pp 249 - 263 Elsevier Scientific Publishing Company.

62. "Evaluation of Tailings Stabilization Methods" by G. M. Ritcey, J. M. Skeaf, V. I. Lakshmannan, and L. M. Luckevich CUPMC , May 1982, Proceedings.

63. "Palabora Mining Company Heavy Minerals Plant Uranium Recovery from Copper Tailings" by R. C. M. van der Spuy Uranium 82. 12th Annual Hydrometallurgy Meeting Toronto Aug 29 to Sept 1, 1982.

64. "Stabilization of Fine Uranium Mill Tailings" by D. Marcus & D. A. Sangrey available from Gulf Research & Development Co. Pittsburgh, PA.

65. "Submission to the Environmental Assessment Board Regarding Uranium Mine Expansion in the Elliot Lake Area" by The Ontario Region of the Environmental Protection Service of the Department of Fisheries and Environment. April 19, 1977.

66. "A Study of Facilities Relative to Stabilization of Uranium Mill Tailings at Elliot Lake" Final Report by Simon-Carves of Canada Ltd. March 31, 1983. Project No. C1903

67. "Report of the National Technical Planning Group on Uranium Tailings Research" to The Canada Centre for Mineral & Energy Technology. (The Lapp Report) September 1981.

68. "The Cluff Lake Board of Inquiry - Final Report" Chairman Hon. Mr. Justice E. D. Bayda, May 31, 1978.

69. "Environmental Assessment of the Proposed Elliot Lake Uranium Mines Expansion" Vol 2, Background Information Update, Feb, 1978.

70. "Environmental Assessment of the Proposed Elliot Lake Uranium Mines Expansion" Vol. 4 Environmental Assessment April, 1978.

71. "The Expansion of the Uranium Mines in the Elliot Lake Area" Final Report, May 1979. Environmental Assessment Board.

APPENDIX C
WATER SAMPLES - ELLIOT LAKE

SECTION C - I
DUNLOP LAKE & EVANS LAKE
1976 & 1983

SAMPLE POINTS Q 14, 15, 19, 20

Samples are taken at the head of the Serpent River, from Dunlop Lake before tailings effluents are added, and from Evans Lake, adjacent to Quirke tailings, draining into Dunlop. Dunlop Lake is "cottage country", and also supplies the potable water for the Quirke operations.

C - I - 1	1983	Dunlop & Evans
C - I - 2	1976	Dunlop Lake - 2 samples.

Note: For a schematic showing location of Sample points, see page C-41.

SECTION C - I - 1

1983 READINGS

DUNLOP LAKE Q - 19

EVANS LAKE Q - 20

16:35: 9 18-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : Q-19 - QUIRKE - DUNLOP LAKE AT PUMPHOUSE

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITH-METIC MEAN	PH GEOM-ETRIC MEAN	ACID-ITY as CaCO3 mg/L	ALKAL-INITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SUL-PHATE SO4 mg/L	RADIUM DISS-OLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MAN-GANESE Mn mg/L	COPPER Cu mg/L
JAN.		6.00	6.00	4	3		35	4		2							
FEB.		5.90	5.90	2	4		40	3		1							
MAR.		6.70	6.70	4	5		35	4		1							
APR.		6.40	6.40	2	3		35	4		1							
MAY		6.15	6.15	4	4	1	23	4	1	2	0.002	0.007	0.001	0.001	0.17	0.020	0.003
JUNE		6.80	6.80	3	4		36	3		2							
JULY		7.03	7.03	1	7		48	4		3							
AUG.		6.64	6.64	1	2	1	21	6		1							
SEPT		6.59	6.59	1	4	9	22	4		1							
OCT.		6.80	6.80	2	3	1	26	4		1							
NOV.		6.50	6.30	1	4	3	22	6	1	1	0.002	0.002	0.001	0.004	0.03	0.011	0.003
DEC.		6.85	6.85	1	4	1	27	1		1							
OF SAMPLES		12	12	12	12	6	12	12	2	12	2	2	2	2	2	2	2
HIGH		7.03	7.03	4	7	9	48	6	1	3	0.002	0.007	0.001	0.004	0.17	0.020	0.003
LOW		5.90	5.90	1	2	1	21	1	1	1	0.002	0.002	0.001	0.001	0.03	0.011	0.003
AVG		6.53	6.52	2	4	3	31	4	1	1	0.002	0.005	0.001	0.003	0.10	0.016	0.003

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : Q-19 - QUIRKE - DUNLOP LAKE AT PUMPHOUSE

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
JAN.					0.60	0.10							1.0
FEB.					0.40	0.08							0.1
MAR.					0.30	0.07							0.5
APR.					0.40	0.09							2.0
MAY		0.530	3.6	0.01	0.20	0.01	0.89			0.01	0.040	0.52	6.5
JUNE					0.20	0.01							21.0
JULY					0.17	0.01							21.0
AUG.					0.20	0.02							24.0
SEPT					0.25	0.05							23.0
OCT.					0.16	0.05							17.0
NOV.		0.520	3.9	0.01	0.20	0.01	0.98				0.010	0.20	9.5
DEC.					0.20	0.04							2.2
# OF SAMPLES		2	2	2	12	12	2			1	2	2	12
HIGH		0.530	3.9	0.01	0.60	0.10	0.98			0.01	0.040	0.52	24.0
LOW		0.520	3.6	0.01	0.16	0.01	0.89			0.01	0.010	0.20	0.1
AVG		0.525	3.8	0.01	0.27	0.05	0.94			0.01	0.025	0.36	10.6

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:35:14 10-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : Q-20 - QUIRKE - EVANS LAKE OUTLET TO DUNLOP LAKE

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.	28.7	6.40	6.40	6	15		39	4		2							
FEB.	17.2	7.00	7.00	4	16		40	4		1							
MAR.	18.5	6.90	6.90	3	17		45	6		1							
APR.	54.5	6.25	6.25	4	14		38	4		1							
MAY	48.5	6.50	6.50	4	15	2	38	3	1	3	0.002	0.007	0.001	0.001	0.11	0.060	0.007
JUNE	24.7	6.30	6.30	5	17		47	4		3							
JULY	13.7	7.30	7.30	2	17		42	4		4							
AUG.	2.9	7.30	7.30	1	17	8	36	3		1							
SEPT	1.3	6.87	6.87	1	18	2	48	3		1							
OCT.	16.9	7.25	7.25	3	16	1	42	4		1							
NOV.	19.0	7.00	7.00	1	16	1	37	4	2	2	0.002	0.002	0.001	0.001	0.05	0.013	0.001
DEC.	20.1	7.30	7.30	1	16	2	34	1		1							
# OF SAMPLES	12	12	12	12	12	6	12	12	2	12	2	2	2	2	2	2	2
HIGH	54.5	7.30	7.30	6	18	8	48	6	2	4	0.002	0.007	0.001	0.001	0.11	0.060	0.007
LOW	1.3	6.25	6.25	1	14	1	34	1	1	1	0.002	0.002	0.001	0.001	0.05	0.013	0.001
AVG	22.2	6.86	6.85	3	16	3	41	4	2	2	0.002	0.005	0.001	0.001	0.08	0.037	0.004

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:35:15 18-JAN-84

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : Q-20 - QUIRKE - EVANS LAKE OUTLET TO DUNLOP LAKE

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNE-SIUM Mg mg/L	CALCIUM Ca mg/L	NIT-RITE NO2 as N mg/L	NIT-RATE NO3 as N mg/L	AMMO-NIA NH3 as N mg/L	KJEL-DAHL N as N mg/L	OIL AND GREASE mg/L	TURB-IDITY JACK-SON UNITS	URANIUM U3O8 mg/L	PHOS-PHORUS P mg/L	CHLORIDE Cl mg/L	TEMPE-RATURE CEL
JAN.	28.7				0.50	0.05							1.0
FEB.	17.2				0.40	0.07							0.5
MAR.	18.5				0.30	0.02							0.5
APR.	54.5				0.45	0.01							1.5
MAY	48.5	0.820	7.0	0.01	0.30	0.09	1.19			0.01	0.040	0.75	8.0
JUNE	24.7				0.30	0.04							22.0
JULY	13.7				0.28	0.01							22.0
AUG.	2.9				0.20	0.05							24.8
SEPT	1.3				0.19	0.05							23.5
OCT.	16.9				0.34	0.03							17.0
NOV.	19.0	1.100	9.5	0.01	0.27	0.01	1.30				0.010	0.20	9.0
DEC.	20.1				0.30	0.01							1.5
# OF SAMPLES	12	2	2	2	12	12	2			1	2	2	12
HIGH	54.5	1.100	9.5	0.01	0.50	0.09	1.30			0.01	0.040	0.75	24.8
LOW	1.3	0.820	7.0	0.01	0.19	0.01	1.19			0.01	0.010	0.20	0.5
AVG	22.2	0.960	8.3	0.01	0.32	0.04	1.25			0.01	0.025	0.48	10.9

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

SECTION C - I - 2

DUNLOP LAKE - 1976

DUNLOP LAKE OUTLET SAMPLE Q - 14

DUNLOP LAKE NEAR GRAVEL PIT SAMPLE Q - 15

WATER SAMPLE DETAILED REPORT-CONCENTRATIONS-YEAR TO DATE

SAMPLE LOCATION: O-15 QUIRKE - DUNLOP L., BAY WHERE CREEK FORM GRAVEL PIT L. ENTERS
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE	FLOW	PH	ACIDI-ITY	ALKA-LINITY	S.S.	T.D.S.	SUL-PHATE	NIT-RATE NO3	AMM-ONIA NH3	RADIUM	URAN-IUM	COPPER	IRON	COBALT	ZINC	MAN-GANESE	LEAD	NICKE
MONTH	IGPM		CACO3 MG/L	CACO3 MG/L	MG/L	MG/L	SO4 MG/L	AS N MG/L	AS N MG/L	RA PC/L	U308 MG/L	CU MG/L	FE MG/L	CO MG/L	ZN MG/L	MN MG/L	PB MG/L	NI MG/L
AUG.	0	6.00	0	0	0	60	0	0.	0.	1.0	0.	0.	0.	0.	0.	0.	0.	0.
SEPT	0	6.85	0	0	0	52	6	0.	0.	1.6	0.	0.	0.	0.	0.	0.	0.	0.
OCT.	0	6.80	0	0	0	48	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
NOV.	0	6.90	0	0	0	24	4	0.	0.	2.0	0.	0.	0.	0.	0.	0.	0.	0.
DEC.	0	6.75	0	0	0	52	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
# OF SAMPLES	0	5	0	0	0	5	2	0	0	3	0	0	0	0	0	0	0	0
HIGH	0	6.90	0	0	0	60	6	0.	0.	2.0	0.	0.	0.	0.	0.	0.	0.	0.
LOW	0	6.00	0	0	0	24	4	0.	0.	1.0	0.	0.	0.	0.	0.	0.	0.	0.
AVERAGE	0	6.66	0	0	0	47	5	0.	0.	1.5	0.	0.	0.	0.	0.	0.	0.	0.

I WATER SAMPLE DETAILED REPORT-CONCENTRATIONS-YEAR TO DATE

SAMPLE LOCATION: 0-14 QUIRKE - DUNLOP LAKE OUTLET
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE MONTH	FLOW IGPM	PH	ACIDITY CACO3 MG/L	ALKALINITY CACO3 MG/L	S.S. MG/L	T.D.S. MG/L	SULPHATE SO4 MG/L	NITRATE HO3 AS N MG/L	AMMONIA NH3 AS N MG/L	RADIUM RA PC/L	URANIUM U308 MG/L	COPPER CU MG/L	IRON FE MG/L	COBALT CO MG/L	ZINC ZN MG/L	MANGANESE MN MG/L	LEAD PB MG/L	NICKE NI MG/L
JAN	0	6.50	0	0	0	48	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
FEB.	0	6.65	0	0	0	56	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
MAR.	0	6.55	0	0	0	32	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
APR.	0	6.40	0	0	0	20	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
MAY	56554	6.60	0	0	0	36	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
JUNE	29000	7.10	0	0	0	4	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
JULY	13200	6.60	0	0	0	16	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
AUG.	3600	6.85	0	0	0	28	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
SEPT	200	6.30	0	0	0	20	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
OCT.	150	6.80	0	0	0	24	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
NOV.	200	6.90	0	0	0	44	5	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
DEC.	200	6.55	0	0	0	20	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
# OF SAMPLES	8	12	0	0	0	12	1	0	0	0	0	0	0	0	0	0	0	0
HIGH	56554	7.10	0	0	0	56	5	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
LOW	150	6.40	0	0	0	4	5	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
AVERAGE	12888	6.71	0	0	0	29	5	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.

APPENDIX C - II - PRONTO

C - II - 1 - COMPARISONS - 1976 & 1983

C - II - 2 - RADIUM RELEASES - 1973 to 1983

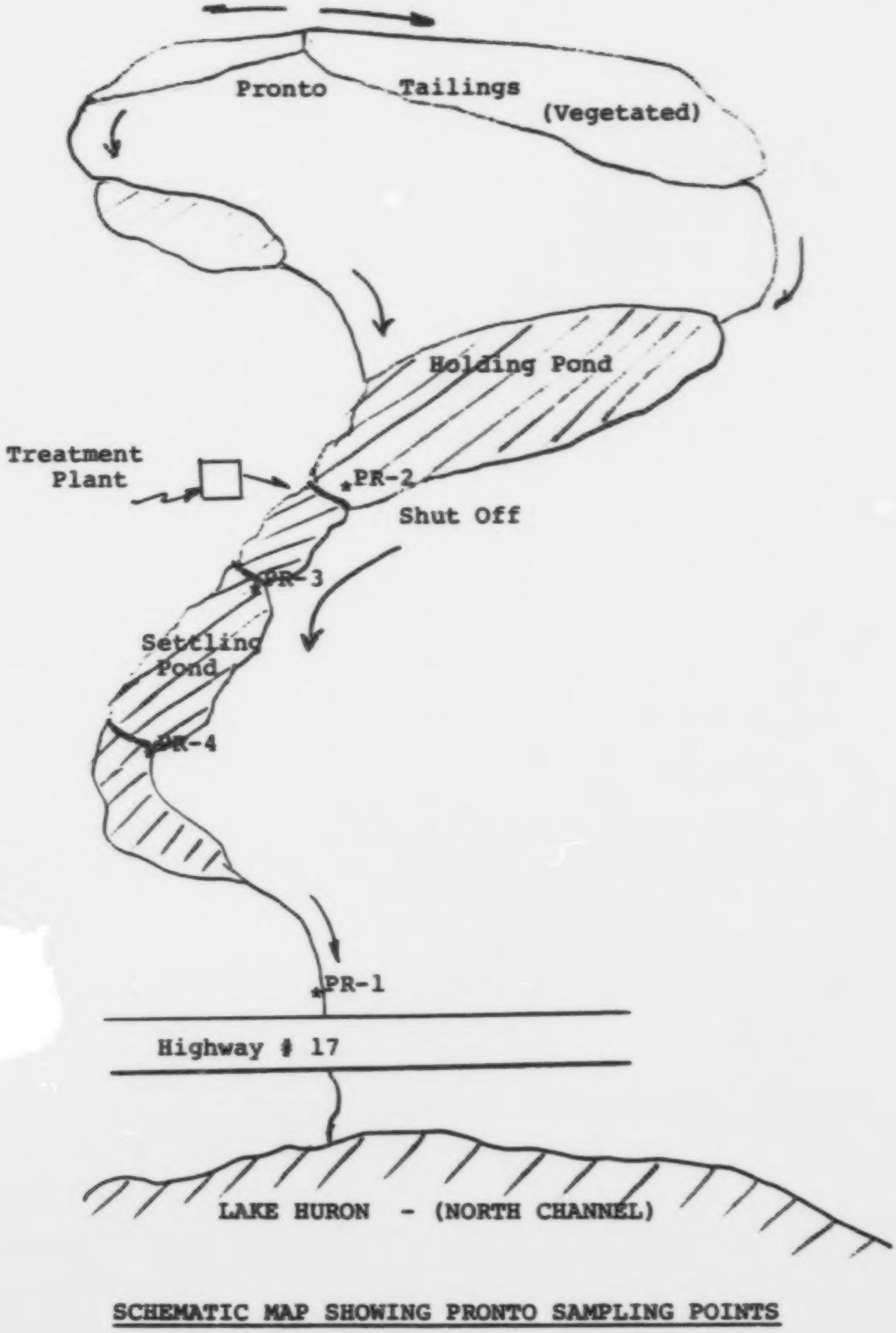
C - II - 1 - READINGS 1976 & 1983

PR - 1 - EFFLUENT AT HIGHWAY 17

PR - 2 - FEED TO PRONTO TREATMENT PLANT

PR - 3 - TREATED EFFLUENT LEAVING TREATMENT PLANT

PR - 4 - TREATED EFFLUENT O/F SETTLING AREA


Pronto Tailings (Vegetated)
Holding Pond
Treatment Plant
PR-2
Shut Off
PR-3
Settling Pond
PR-4
PR-1
Highway # 17
LAKE HURON - (NORTH CHANNEL)

SCHEMATIC MAP SHOWING PRONTO SAMPLING POINTS

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION: PR-1 PRONTO - EFFLUENT AT HIGHWAY 17
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE MONTH	FLOW IGPM	PH	ACIDITY CACO3 MG/L	ALKALINITY CACO3 MG/L	S.S. MG/L	T.D.S. MG/L	SULPHATE SO4 MG/L	NITRATE NO3 AS N MG/L	AMMONIA NH3 AS N MG/L	RADIUM RA PC/L	URANIUM U3O8 MG/L	COPPER CU MG/L	IRON FE MG/L	COBALT CO MG/L	ZINC ZN MG/L	MANGANESE MN MG/L	LEAD PB MG/L	NICKE NI MG/L
JAN.	18	6.13	0	0	0	604	0	0.	0.	1.0	0.	.04	.81	.13	.05	.49	.05	.03
FEB.	20	6.43	0	0	0	576	352	0.	0.	4.7	0.	.04	1.30	.12	.02	.64	.06	.03
MAR.	859	6.33	0	0	0	516	0	0.	0.	5.7	0.	.03	1.64	.07	.01	.69	.07	.04
APR.	1831	8.98	0	0	0	488	0	0.	0.	3.3	0.	.02	.58	.03	.01	.05	.06	.04
MAY	997	6.98	3	14	0	529	303	.80	.80	1.2	0.	.02	.26	.03	.01	.04	.07	.03
JUNE	19	6.95	0	0	0	145	0	0.	0.	6.3	0.	.02	2.70	.02	.03	.28	.05	.03
JULY	3	7.16	0	0	0	316	0	0.	0.	7.8	0.	.02	3.40	.03	.02	.23	.04	.03
AUG.	1	7.32	0	0	0	309	40	0.	0.	12.6	0.	.02	3.50	.02	.04	.51	.04	.02
SEPT	1	7.11	0	0	0	374	0	0.	0.	10.0	0.	.02	.91	.02	.02	.37	.04	.02
OCT.	60	6.72	0	0	0	787	0	0.	0.	6.6	0.	.02	.74	.03	.01	.16	.05	.03
NOV.	403	6.77	0	0	0	1159	747	0.	0.	5.3	0.	.02	12.00	.08	.03	.28	.04	.03
DEC.	36	6.58	0	0	0	1344	0	0.	0.	7.6	0.	.02	1.40	.05	.03	.37	.05	.03
# OF SAMPLES	12	12	1	1	0	12	4	1	1	12	0	12	12	12	12	12	12	12
HIGH	1831	8.98	3	14	0	1344	747	.8	.8	12.6	0.	.04	12.00	.13	.05	.69	.07	.04
LOW	1	6.13	3	14	0	145	40	.8	.8	1.0	0.	.02	.26	.02	.01	.04	.04	.02
AVERAGE	354	6.95	3	14	0	602	360	.8	.8	6.0	0.	.02	2.44	.05	.02	.34	.05	.03

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION: PR-2 PRONTO - FEED TO PRONTO TREATMENT PLANT
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE MONTH	FLOW IGPM	PH	ACIDITY CACO3 MG/L	ALKALINITY CACO3 MG/L	S.S. MG/L	T.D.S. MG/L	SULPHATE SO4 MG/L	NITRATE NO3 AS N MG/L	AMMONIA NH3 AS N MG/L	RADIUM RA PC/L	URANIUM U3O8 MG/L	COPPER CU MG/L	IRON FE MG/L	COBALT CO MG/L	ZINC ZN MG/L	MANGANESE MN MG/L	LEAD PB MG/L	NICKE NI MG/L
JAN.	5	3.09	179	0	0	0	0	0.	0.	18.3	0.	.75	24.00	.61	.15	1.10	.05	.08
FEB.	1	3.38	216	0	0	1347	0	0.	0.	28.4	0.	1.80	51.00	.89	.26	1.70	.07	.16
MAR.	1169	3.70	173	0	0	795	0	0.	0.	21.6	0.	.76	37.00	.55	.13	1.33	.07	.09
APR.	1977	3.31	149	0	0	802	0	0.	0.	20.5	0.	.71	32.00	.68	.14	1.19	.06	.08
MAY	635	2.94	211	0	0	728	500	3.60	.60	22.2	0.	.87	50.00	.59	.13	1.10	.07	.09
JUNE	1	2.85	195	0	0	345	0	0.	0.	29.0	0.	.68	12.00	.65	.14	1.40	.05	.10
JULY	1	2.84	182	0	0	981	0	0.	0.	38.0	0.	.66	10.00	.58	.16	1.63	.04	.10
AUG.	1	2.95	180	0	0	1145	0	0.	0.	28.1	0.	.59	7.80	.53	.14	2.00	.04	.11
SEPT	1	3.17	172	0	0	1302	0	0.	0.	26.5	0.	.64	6.70	.55	.15	1.90	.06	.11
OCT.	242	3.09	219	0	0	1265	0	0.	0.	49.1	0.	.49	27.00	.76	.19	2.20	.06	.12
NOV.	421	2.87	298	0	0	1346	0	0.	0.	44.8	0.	.58	48.00	.77	.18	2.30	.06	.13
DEC.	0	2.92	301	0	0	0	0	0.	0.	42.1	0.	1.00	46.00	.92	.27	2.60	.06	.14
# OF SAMPLES	11	12	12	0	0	10	1	1	1	12	0	12	12	12	12	12	12	12
HIGH	1977	3.70	301	0	0	1347	500	3.6	.6	49.1	0.	1.80	51.00	.92	.27	2.60	.07	.16
LOW	1	2.84	149	0	0	345	500	3.6	.6	18.3	0.	.49	6.70	.53	.13	1.10	.04	.08
AVERAGE	405	3.09	206	0	0	1005	500	3.6	.6	30.7	0.	.79	29.29	.67	.17	1.70	.06	.11

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION: PR-3 PRONTO - TREATED EFFLUENT LEAVING TREATMENT PLANT
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE MONTH	FLOW IGPM	PH	ACIDITY CACO3 MG/L	ALKALINITY CACO3 MG/L	S.S. MG/L	T.D.S. MG/L	SULPHATE SO4 MG/L	NITRATE NO3 AS N MG/L	AMMONIA NH3 AS N MG/L	RADIUM RA PC/L	URANIUM U3O8 MG/L	COPPER CU MG/L	IRON FE MG/L	COBALT CO MG/L	ZINC ZN MG/L	MANGANESE MN MG/L	LEAD PB MG/L	NICKE NI MG/L
MAR.	0	10.20	0	0	0	0	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
APR.	0	9.43	0	0	0	0	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
MAY	0	9.88	0	0	0	0	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
OCT.	0	10.27	0	0	0	0	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
NOV.	0	10.14	0	0	0	0	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
# OF SAMPLES	0	5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
HIGH	0	10.27	0	0	0	0	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
LOW	0	9.43	0	0	0	0	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
AVERAGE	0	9.98	0	0	0	0	0	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.

I WATER SAMPLE DETAILED REPORT-CONCENTRATIONS-YEAR TO DATE

SAMPLE LOCATION: PR-4 PRONTO - TREATED EFFLUENT O/F SETTLING AREA
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE MONTH	FLOW IGPM	PH	ACIDI- ITY CACO3 MG/L	ALKA- LINITY CACO3 MG/L	S.S. MG/L	T.D.S. MG/L	SUL- PHATE SO4 MG/L	NIT- RATE NO3 AS N MG/L	AMM- ONIA NH3 AS N MG/L	RADIUM RA PC/L	URAN- IUM U3O8 MG/L	COPPER CU MG/L	IRON FE MG/L	COBALT CO MG/L	ZINC ZN MG/L	MAN- GANESE MN MG/L	LEAD PB MG/L	NICKE NI MG/L
MAR.	0	10.50	0	0	0	0	0	0.	0.	6.4	0.	0.	0.	0.	0.	0.	0.	0.
APR.	0	9.76	0	0	0	0	0	0.	0.	4.7	0.	0.	0.	0.	0.	0.	0.	0.
MAY	0	9.78	3	15	0	712	456	.30	.60	5.6	0.	.02	.24	.03	.01	.01	.07	.03
NOV.	0	6.72	0	0	0	0	0	0.	0.	7.4	0.	.09	3.85	.20	.05	.63	.04	.04
# OF SAMPLES	0	4	1	1	0	1	1	1	1	4	0	2	2	2.	2	2	2	2
HIGH	0	10.50	3	15	0	712	456	.3	.6	7.4	0.	.09	3.85	.20	.05	.63	.07	.04
LOW	0	6.72	3	15	0	712	456	.3	.6	4.7	0.	.02	.24	.03	.01	.01	.04	.03
AVERAGE	0	9.19	3	15	0	712	456	.3	.6	6.0	0.	.06	2.05	.11	.03	.32	.05	.04

16:33:17 10-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : PR-01 - PRONTO - EFFLUENT AT HWY 17

FOR YEAR 1983
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TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.	2.1	6.27	6.27														
FEB.	2.8	6.57	6.57	20	57		429	180		1	0.012	0.140	0.008	0.043	0.37	0.550	0.013
MAR.	200.2	9.81	9.74	2	20		325		1		0.003	0.020	0.010	0.020	0.41	0.120	0.009
APR.	170.3	8.29	8.22	N	28		278			1	0.005	0.019	0.008	0.008	0.24	0.042	0.005
MAY	84.3	7.06	7.06	5	23	11	549	204	1	1	0.002	0.004	0.001	0.008	0.15	0.001	0.007
JUNE	84.1	7.04	7.04	5	40		545			11	0.006	0.005	0.005	0.012	0.16	0.058	0.006
JULY	12.1	7.24	7.24	4	46		728			2	0.002	0.003	0.010	0.011	0.52	0.110	0.008
AUG.	9.5	6.76	6.76	6	39	1	818	502		3	0.002	0.003	0.001	0.002	0.16	0.043	0.005
SEPT	6.5	6.83	6.83	7	64	2	875			2	0.013	0.017	0.016	0.017	0.16	0.060	0.006
OCT.	41.0	6.71	6.71														
NOV.	92.6	7.14	7.10	6	28	5	740	467		1	0.016	0.005	0.004	0.099	0.32	0.016	0.007
DEC.	64.1	7.35	7.30	N	20	2	841			1	0.014	0.003	0.009	0.011	0.06	0.013	0.006
- OF SAMPLES	12	12	12	10	10	5	10	4	2	9	10	10	10	10	10	10	10
HIGH	200.2	9.81	9.74	20	64	11	875	507	1	11	0.016	0.140	0.016	0.099	0.57	0.550	0.013
LOW	2.1	6.27	6.27	N	20	1	278	180	1	1	0.002	0.003	0.001	0.002	0.06	0.001	0.005
AVG	64.1	7.26	7.19	6	37	4	615	338	1	3	0.008	0.072	0.007	0.023	0.25	0.103	0.007

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:33:18 18-JAN-84

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : PR-01 - PRONTO - EFFLUENT AT HWY 17

FOR YEAR 1983
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TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
MAY	84.3	6.000	84.0	0.01	0.50	0.50	1.50			0.01	0.010	29.00	
# OF SAMPLES	1	1	1	1	1	1	1			1	1	1	
HIGH	84.3	6.000	84.0	0.01	0.50	0.50	1.50			0.01	0.010	29.00	
LOW	84.3	6.000	84.0	0.01	0.50	0.50	1.50			0.01	0.010	29.00	
AVG	84.3	6.000	84.0	0.01	0.50	0.50	1.50			0.01	0.010	29.00	

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:33:25 10-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : PR-02 - PRONTO - FEED TO PRONTO TREATMENT PLANT BEFORE CAUSEWAY

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.		3.82	3.82	65													
FEB.		4.80	4.54	39			227	131		2	0.012	0.026	0.020	0.075	4.90	0.170	0.080
MAR.	135.3	6.27	5.98	32	10		110		1		0.003	0.030	0.007	0.040	1.60	0.110	0.030
APR.	114.0	3.64	3.64	98			387			4	0.011	0.049	0.034	0.180	11.00	0.450	0.144
MAY	54.0	3.36	3.36	102	N	9	604	457	5	8	0.010	0.042	0.036	0.280	27.00	0.750	0.230
JUNE	44.1	3.18	3.18	126			378			13	0.016	0.040	0.032	0.280	10.00	0.720	0.170
JULY	32.6	3.02	3.02	121			882			9	0.009	0.040	0.044	0.280	8.90	0.940	0.120
AUG.	55.5	2.95	2.95	155		1	920			13	0.020	0.036	0.039	0.270	10.00	0.930	0.093
SEPT	4.8	3.06	3.06	140		1	1001			17	0.020	0.042	0.050	0.300	9.00	1.200	0.130
OCT.		3.54	3.54	69													
NOV.	109.5	3.47	3.47	75		10	924			12	0.023	0.060	0.058	0.530	4.20	0.740	0.390
DEC.	24.2	4.16	4.14	54		5	789			3	0.013	0.057	0.068	0.490	3.90	0.570	0.370
# OF SAMPLES	9	12	12	12	2	5	10	2	2	9	10	10	10	10	10	10	10
HIGH	135.3	6.27	5.98	155	10	10	1001	457	5	17	0.023	0.060	0.068	0.530	27.00	1.200	0.390
LOW	4.8	2.95	2.95	32	N	1	110	131	1	2	0.003	0.026	0.007	0.040	1.60	0.110	0.030
AVG	63.8	3.77	3.65	90	5	5	622	294	3	9	0.014	0.042	0.039	0.273	9.05	0.658	0.176

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : PR-02 - PRONTO - FEED TO PRONTO TREATMENT PLANT BEFORE CAUSEWAY

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
MAY	54.0	36.000	88.0	0.03	1.20	1.20	2.10			0.25	0.010	15.00	
# OF SAMPLES	1	1	1	1	1	1	1			1	1	1	
HIGH	54.0	36.000	88.0	0.03	1.20	1.20	2.10			0.25	0.010	15.00	
LOW	54.0	36.000	88.0	0.03	1.20	1.20	2.10			0.25	0.010	15.00	
AVG	54.0	36.000	88.0	0.03	1.20	1.20	2.10			0.25	0.010	15.00	

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

-

12:55:56 19-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : PR-03 - PRONTO - TREATED EFFLUENT LEAVING TREATMENT PLANT

FOR YEAR 1983
PAGE 1 OF 1 PAGE ONLY

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/l	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
MAR.		10.79	10.79														
APR.		10.30	10.27														
MAY		11.41	11.41														
JUNE		10.76	10.73														
JULY		11.13	11.11														
AUG.		10.33	10.31														
SEPT		10.97	10.74														
NOV.		10.50	10.49														
DEC.		9.90	9.90														
# OF SAMPLES		9	9														
HIGH		11.41	11.41														
LOW		9.90	9.90														
AVG		10.67	10.63														

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:33:41 18-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : PR-04 - PRONTO - TREATED EFFLUENT O/F SETTLING AREA

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITH-METIC MEAN	PH GEOM-ETRIC MEAN	ACID-ITY as CaCO3 mg/l.	ALKAL-INITY as CaCO3 mg/l.	S.S. mg/L	T.D.S. mg/L	SUL-PHATE SO4 mg/L	RADIUM DISS-OLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MAN-GANESE Mn mg/L	COPPER Cu mg/L
JAN.		8.20	8.20														
MAR.		9.54	9.14			10			2	2	0.003	0.010	0.007	0.020	0.61	0.060	0.010
APR.		9.38	9.12			24			2	3	0.013	0.058	0.005	0.038	1.70	0.088	0.020
MAY		10.73	10.71	N	59	4	608	369	1	2	0.008	0.004	0.010	0.007	0.12	0.007	0.006
JUNE		10.68	10.67			27			2	3	0.020	0.004	0.006	0.012	0.04	0.005	0.005
JULY		10.40	10.40			33			2	3	0.002	0.006	0.010	0.020	0.92	0.045	0.009
AUG.		10.02	10.01			19			4	4	0.015	0.003	0.013	0.002	0.06	0.044	0.006
SEPT		8.82	8.76			6			2	3	0.013	0.006	0.012	0.013	0.12	0.071	0.008
NOV.		9.95	9.87			5			1	2	0.023	0.003	0.010	0.019	0.10	0.015	0.010
DEC.		9.76	9.76			9			1	2	0.013	0.003	0.010	0.029	0.17	0.045	0.014
# OF SAMPLES		10	10	1	1	9	1	1	9	9	9	9	9	9	9	9	9
HIGH		10.73	10.71	N	59	33	608	369	4	4	0.023	0.058	0.013	0.038	1.70	0.088	0.020
LOW		8.20	8.20	N	59	4	608	369	1	2	0.002	0.003	0.005	0.002	0.04	0.005	0.005
AVG		9.75	9.63	N	59	15	608	369	2	2	0.012	0.011	0.009	0.018	0.43	0.042	0.010

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:33:42 18-JAN-84

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : PR-04 - PRONTO - TREATED EFFLUENT O/F SETTLING AREA

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
MAY		5.600	171.0	0.01	0.10	0.80	1.90			0.01	0.010	15.00	
# OF SAMPLES		1	1	1	1	1	1			1	1	1	
HIGH		5.600	171.0	0.01	0.10	0.80	1.90			0.01	0.010	15.00	
LOW		5.600	171.0	0.01	0.10	0.80	1.90			0.01	0.010	15.00	
AVG		5.600	171.0	0.01	0.10	0.80	1.90			0.01	0.010	15.00	

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

APPENDIX C - II - 2

PRONTO RADIUM 1973 TO 1983

VALUES ENTERING & LEAVING TREATMENT PLANT

SAMPLE POINTS : PR - 1 - LEAVING AREA FOR LAKE HURON
PR - 2 - ENTERING TREATMENT PLANT AREA

DATE	PR-2 Entering			$BaCl_2.2H_2O$ Added		PR-1 Leaving	
	Vol kt/mo	Ra pc/L	pH	kg/mo	mg/L	pH	Ra pc/L
1973							
Jan	69	-	-	100	1.4	6.98	5
Feb	94	-	-	100	1.1	7.91	12
Mar	237	-	-	182	0.8	9.95	1
Apr	182	-	-	100	0.6	7.18	5
May	199	21	-	475	2.4	7.48	3
June	100	-	-	175	1.8	7.40	2
July	-	-	-	-	-	6.91	23
Aug	209	-	-	275	1.3	7.24	9
Sept	-	-	-	-	-	6.98	7
Oct	87	-	-	100	1.1	7.17	3
Nov	82	28	3.25	113	1.4	6.73	1
Dec	75	-	-	50	0.7	6.20	3
Total	1334			1670			
Average		24.5					6.17
1974							
Jan	11	-	3.80	23	1.3	6.78	5
Feb	81	22	3.20	150	1.9	7.95	5
Mar	10	20	3.34	25	2.5	6.80	15
Apr	476	9	3.83	350	0.7	8.69	7
May	153	23	3.03	425	2.8	6.91	2
June	137	23	3.06	375	2.7	6.78	3
July	74	36	3.02	325	4.4	6.93	9
Aug	-	41	2.94	-	-	7.28	7
Sept	-	40	2.91	-	-	7.08	11
Oct	49	36	3.24	225	4.5	6.37	2
Nov	205	24	3.28	875	4.3	6.93	1
Dec	36	22	3.32	250	7.0	6.53	3
Total	1232			3023			
Average		26.91					5.83

DATE	PR-2 Entering			$BaCl_2.2H_2O$ Added		PR-1 Leaving	
	Vol kt/mo	Ra pc/L	pH	kg/mo	mg/L	pH	Ra pc/L
1975							
Jan	-	22	3.47	-	-	6.30	1
Feb	49	21	3.32	200	4	7.60	1.3
Mar	118	26	3.24	250	2.1	5.70	3.3
Apr	358	4	3.43	900	2.5	7.00	1.2
May	224	16	3.20	950	4.2	7.14	1
June	109	21	2.98	650	5.9	6.61	1
July	31	28	3.00	150	4.8	7.18	4.7
Aug	-	34	2.98	-	-	7.14	2
Sept	104	50	3.03	750	7.2	6.93	3
Oct	-	21	3.05	-	-	6.35	7
Nov	31	16	3.06	250	8.2	6.64	1.8
Dec	161	10	3.17	650	4	5.82	2.1
Total	1185			4750			
Average		22.42					2.45
1976							
Jan	-	18	3.09	-	-	6.13	1
Feb	-	28	3.38	-	-	6.43	5
Mar	237	22	3.70	750	3.8	6.33	6
Apr	388	21	3.31	2000	5.2	8.98	3
May	129	22	2.94	950	7.4	6.97	1
June	-	29	2.85	-	-	6.95	6
July	-	38	2.84	-	-	7.16	8
Aug	-	28	2.95	-	-	7.32	13
Sept	-	27	3.17	-	-	7.11	10
Oct	29	49	3.09	250	8.6	6.72	7
Nov	49	45	2.87	350	7.1	6.77	5
Dec	-	42	2.92	-	-	6.58	8
Total	832			4300			
Average		30.75					6.08

	PR-2 Entering			$BaCl_2.2H_2O$	Added	PR-1 Leaving	
DATE	Vol kt/mo	Ra pc/L	pH	kg/mo	mg/L	pH	Ra pc/L
1977							
Jan	-	27	2.88	-	-	6.64	5
Feb	-	28	3.09	-	-	7.01	5
Mar	731	21	3.28	1450	3	7.35	6
Apr	326	9	3.37	1450	4	9.37	7
May	167	25	2.95	700	6	7.22	6
June	-	28	2.90	-	-	7.01	8
July	-	5	2.94	-	-	6.89	2
Aug	-	26	3.02	-	-	6.96	4
Sept	58	-	3.05	500	31	6.70	-
Oct	231	17	3.09	1000	4	6.69	7
Nov	370	14	3.08	775	2	6.54	5
Dec	178	7	3.26	700	4	10.11	7
Total	2061			6575			
Average		18.82					5.64
1978							
Jan	139	15	3.18	500	4	7.80	5
Feb	-	3	3.20	-	-	6.83	6
Mar	-	27	3.40	-	-	7.16	8
Apr	319	9	3.47	1575	4.5	9.82	4
May	284	17	3.12	1550	5.5	9.05	5
June	161	22	2.92	900	6.2	7.44	4
July	-	28	3.01	-	-	7.28	13
Aug	-	31	3.30	-	-	7.24	21
Sept	134	22	2.88	450	7	6.98	10
Oct	198	28	3.02	1200	6	6.98	3
Nov	217	20	3.23	1000	4.6	7.55	4
Dec	175	22	3.20	300	3.5	7.25	3
Total	1627			7475			
Average		20.33					7.17

DATE	PR-2 Entering Vol kt/mo	 Ra pc/L	 pH	$BaCl_2.2H_2O$ kg/mo	Added mg/L	PR-1 Leaving pH	 Ra pc/L
1979							
Jan	-	32	3.15	-	-	6.95	4
Feb	-	23	3.35	-	-	6.73	4
Mar	502	14	3.81	1200	2.4	8.46	14
Apr	484	8	3.82	1450	3.0	9.66	5
May	282	17	3.20	950	3.4	7.14	3
June	109	24	2.99	700	6.4	7.11	4
July	-	23	3.06	450	-	6.71	4
Aug	-	42	3.30	-	-	6.87	4
Sept	-	29	3.20	-	-	7.00	17
Oct	-	30	5.92	-	-	6.86	4
Nov	341	15	3.40	1300	3.8	7.05	2
Dec	45	19	3.14	300	6.7	6.14	2
Total	1763			6350			
Average		23					5.58
1980							
Jan	-	4	3.51	-		6.70	2
Feb	-	7	3.25	-	-	6.40	4
Mar	42	8	3.33	400	9.5	6.78	4
Apr	332	3	3.96	1500	4.5	6.96	2
May	251	10	3.50	1400	5.6	6.75	2
June	210	24	3.18	1700	8.1	7.24	8
July	14	25	3.00	550	39.0	6.92	4
Aug	-	34	2.95	-	-	7.06	4
Sept	138	33	3.00	1400	10.1	6.78	7
Oct	125	24	3.08	2000	16	6.49	6
Nov	-	19	3.35	-	-	6.58	3
Dec	-	17	3.22	-	-	6.68	4
Total	1112			8950			
Average		17.33					4.17

DATE	PR-2 Entering			$BaCl_2.2H_2O$ Added		PR-1 Leaving	
	Vol kt/mo	Ra pc/L	pH	kg/mo	mg/L	pH	Ra pc/L
1981							
Jan	-	11	3.36	-	-	6.62	6
Feb	-	16	4.06	-	-	6.05	3
Mar	256	3	4.13	1350	5.3	6.95	2
Apr	289	-	3.60	1550	5.4	8.00	-
May	254	9	3.33	1400	5.5	7.24	3
June	122	15	3.17	1300	10.7	6.26	3
July	76	22	3.06	1600	20.9	6.72	5
Aug	2	14	3.12	150	97.9	7.20	2
Sept	-	24	2.88	-	-	7.38	3
Oct	-	29	3.00	-	-	6.98	3
Nov	57	18	2.98	950	16.7	7.01	2
Dec	69	17	3.06	700	10.1	6.17	3
Total	1125			9000			
Average		16.18					3.18
1982							
Jan	-	14	2.89	-	-	6.37	4
Feb	-	13	2.91	-	-	6.64	3
Mar	98	19	3.60	500	5.1	6.73	5
Apr	393	2	5.08	1150	2.9	7.23	1
May	307	4	3.49	900	2.9	7.14	3
June	147	11	3.05	650	4.4	6.87	2
July	-	17	2.95	-	-	7.28	1
Aug	-	-	2.85	-	-	7.10	-
Sept	113	11	3.00	300	2.7	6.64	3
Oct	287	11	3.08	700	2.4	6.68	3
Nov	259	12	3.13	800	2.7	7.11	3
Dec	217	7	3.51	400	1.8	6.73	2
Total	1821			5400			
Average		11					2.73

DATE	PR-2 Entering			$BaCl_2.2H_2O$ Added		PR-1 Leaving	
	Vol kt/mo	Ra pc/L	pH	kg/mo	mg/L	pH	Ra pc/L
1983							
Jan	-	-	3.83	-	-	6.28	-
Feb	-	2	4.80	-	-	6.57	1
Mar	362	1	6.28	1100	3.0	9.81	1
Apr	296	4	3.64	600	2.0	8.29	1
May	145	8	3.36	650	4.5	7.06	1
June	114	13	3.18	500	4.4	7.04	11
July	87	9	3.02	500	5.7	7.24	2
Aug	149	13	2.95	650	4.4	6.76	3
Sept	12	17	3.06	250	20.1	6.83	2
Oct	-	-	3.54	-	-	6.72	-
Nov	284	12	3.47	1050	3.7	7.14	1
Dec	65	3	4.16	100	1.5	7.35	1
Total	1514			5400			
Average		8.2					2.4

APPENDIX C - III

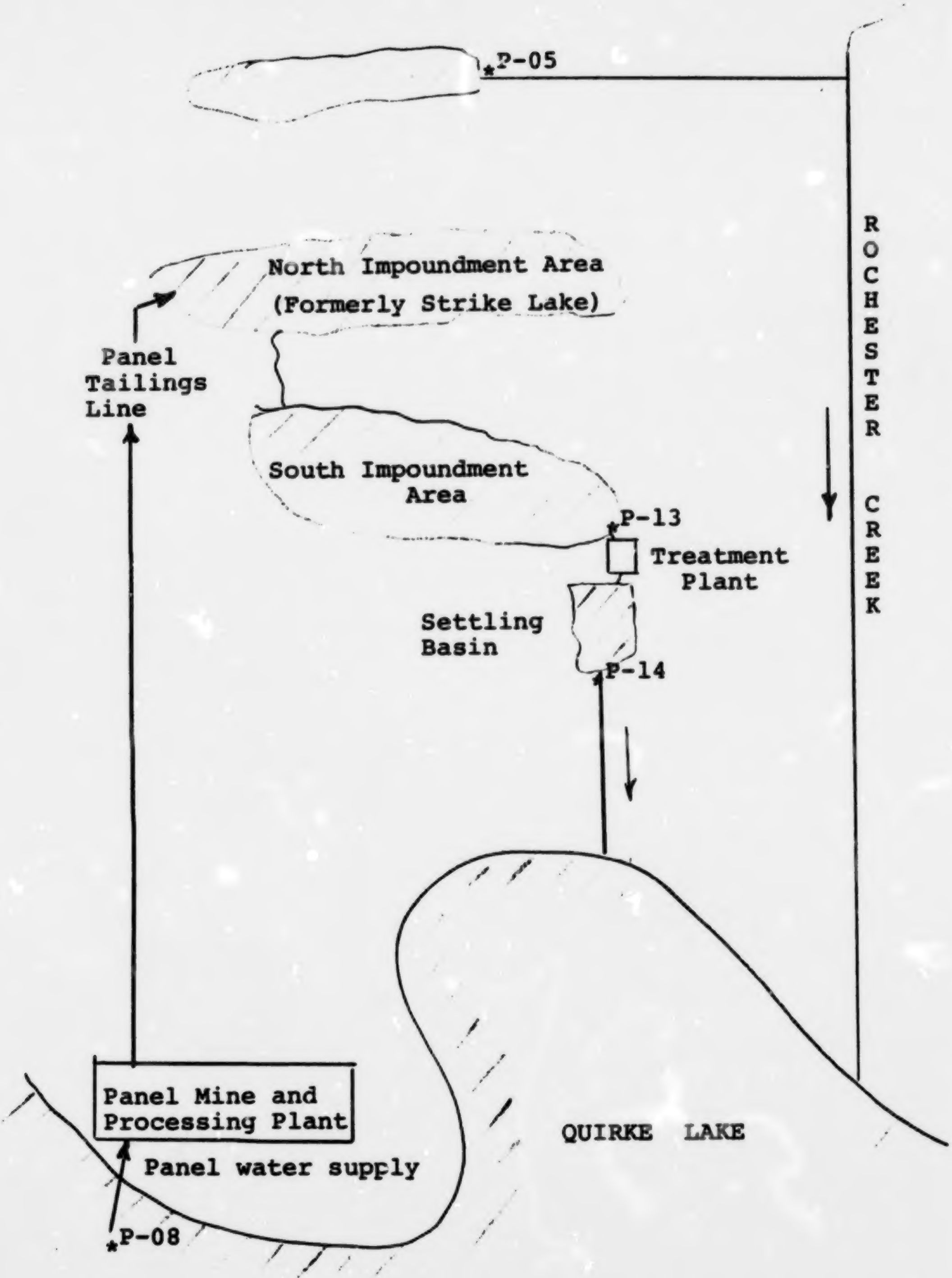
PANEL DATA - 1983

FRESH WATER ABOVE TAILINGS POND - P - 05

QUIRKE LAKE AT PUMPHOUSE - P - 08

FEED TO TREATMENT PLANT - P - 13

TREATED EFFLUENT - P - 14


P-05
ROCHESTER CREEK
North Impoundment Area
(Formerly Strike Lake)
Panel Tailings Line
South Impoundment Area
P-13
Treatment Plant
Settling Basin
P-14
Panel Mine and Processing Plant
Panel water supply
P-08
QUIRKE LAKE

SCHEMATIC SHOWING PANEL SAMPLES

- C-33 -

FRESH – PROVE [illegible]

16:32: 7 18-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : P-05 - PANEL - SWAMP OUTLET NORTHEAST OF STRIKE LAKE

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
FEB.		6.38	6.38	4	15		50			1					0.27		
MAY		5.30	5.30	6	2	1	34	4	6	8	0.022	0.015	0.003	0.001	0.11	0.005	0.011
AUG.		6.55	6.55	4	16	1	49			2					3.10		
NOV.		6.10	6.10	4	2	1	39			1					0.14		
# OF SAMPLES		4	4	4	4	3	4	1	1	4	1	1	1	1	4	1	1
HIGH		6.55	6.55	6	16	1	50	4	6	8	0.022	0.015	0.003	0.001	3.10	0.005	0.011
LOW		5.30	5.30	4	2	1	34	4	6	1	0.022	0.015	0.003	0.001	0.11	0.005	0.011
AVG		6.08	6.06	5	9	1	43	4	6	3	0.022	0.015	0.003	0.001	0.91	0.005	0.011

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:32: 8 18-JAN-84

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : P-05 - PANEL - SWAMP OUTLET NORTHEAST OF STRIKE LAKE

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
MAY		0.350	2.7	0.01	0.60	0.20	0.90			0.01	0.070	0.23	
# OF SAMPLES		1	1	1	1	1	1			1	1	1	
HIGH		0.350	2.7	0.01	0.60	0.20	0.90			0.01	0.070	0.23	
LOW		0.350	2.7	0.01	0.60	0.20	0.90			0.01	0.070	0.23	
AVG		0.350	2.7	0.01	0.60	0.20	0.90			0.01	0.070	0.23	

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:32:11 18-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : P-08 - PANEL - QUIRKE LAKE AT PANEL PUMPHOUSE

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.		6.20	6.20	5	2		319	161	4		0.002	0.016	0.006	0.003	0.17	0.220	0.007
FEB.		6.70	6.70	7	3		357	185	3		0.003	0.040	0.010	0.007	0.06	0.270	0.004
MAR.		6.60	6.60	2	3		167	71	2		0.003	0.110	0.002	0.002	0.38	0.110	0.061
APR.		6.30	6.26	5	4		175	77	5		0.008	0.031	0.024	0.005	0.11	0.113	0.004
MAY		5.70	5.70	6	3		342	173	3		0.004	0.021	0.008	0.001	0.08	0.230	0.015
JUNE		5.80	5.80	6	3		310	164	2		0.002	0.013	0.005	0.006	0.08	0.210	0.005
JULY		6.65	6.65	3	3		343	186	2		0.002	0.014	0.004	0.007	0.07	0.230	0.005
AUG.		6.20	6.20	3	3		379	194	3		0.002	0.011	0.010	0.012	0.06	0.240	0.002
SEPT		6.42	6.42	4	2		376	187	2		0.005	0.011	0.007	0.004	0.02	0.210	0.003
OCT.		6.10	6.10	4	1		364	213	2		0.014	0.013	0.010	0.015	0.05	0.190	0.006
NOV.		6.00	6.00	4	2		362	197	2		0.005	0.013	0.006	0.006	0.05	0.220	0.004
DEC.		6.60	6.60	3	4		348	175	2		0.001	0.014	0.008	0.007	0.10	0.190	0.001
# OF SAMPLES		12	12	12	12		12	12	12		12	12	12	12	12	12	12
HIGH		6.70	6.70	7	4		379	213	5		0.014	0.110	0.024	0.015	0.38	0.270	0.015
LOW		5.70	5.70	2	1		167	71	2		0.001	0.011	0.002	0.001	0.02	0.110	0.001
AVG		6.27	6.26	4	3		320	165	3		0.004	0.026	0.008	0.006	0.10	0.203	0.005

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:32:12 18-JAN-84

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : P-08 - PANEL - QUIRKE LAKE AT PANEL PUMPHOUSE

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
JAN.		2.100		0.13	9.40	4.50	8.20				0.010		
FEB.		2.100		0.02	8.10	4.00	5.10				0.020		
MAR.		1.100		0.50	2.90	4.00	5.10				0.010		
APR.		0.290	38.7	0.04	4.30	2.20	3.25				0.025		
MAY		1.800	73.0		8.60	3.80	4.40				0.060		
JUNE		2.200	63.0	0.15	6.90	3.80	4.80				0.200		
JULY		2.600	69.0	0.13	9.36	4.14	4.59				0.030		
AUG.		2.300	71.0	0.15	9.70	4.60	4.90				0.010		
SEPT		2.800	75.0	0.15	10.60	4.37	4.93				0.030		
OCT.		2.500	73.0	0.12	10.00	4.70	5.10				0.040		
NOV.		2.600	73.0	0.12	10.00	4.50	5.30				0.020		
DEC.		2.600	76.0	0.01	9.90	3.70	4.00				0.010		
# OF SAMPLES		12	9	11	12	12	12				12		
HIGH		2.800	76.0	0.50	10.60	4.70	8.20				0.200		
LOW		0.290	38.7	0.01	2.90	2.20	3.25				0.010		
AVG		2.083	68.0	0.14	8.31	4.03	4.97				0.039		

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

5:57:59 20-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : P-13 - PANEL - TREATMENT AREA FEED

FOR YEAR 1983
PAGE 1 OF 1 PAGE ONLY

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.	125.9	4.96	4.95	16	N					174							
FEB.	129.5	5.20	5.19	18	1					201							
MAR.	110.5	5.38	5.37	20	1					193							
APR.	122.4	5.31	5.31	18	1					198							
MAY	149.9	5.42	5.42	12	3					152							
JUNE	142.0	5.33	5.32	14	1					176							
JULY	90.8	5.61	5.61	11	2					177							
AUG.	56.6	5.61	5.61	9	1	2				139							
SEPT	52.6	5.60	5.59	11	1	2				174							
OCT.	82.9	5.04	5.04	16	0	3				184							
NOV.	75.2	5.46	5.32	19	8	3				171							
DEC.	99.2	4.83	4.81	20	0	5				222							
# OF SAMPLES	12	12	12	12	12	5				12							
HIGH	149.9	5.61	5.61	20	8	5				222							
LOW	52.6	4.83	4.81	9	N	2				139							
AVG	103.1	5.31	5.29	15	2	3				180							

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

12:51:53 19-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : P-14 - PANEL - TAILINGS POND TREATED EFFLUENT

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/l.	ALKALINITY as CaCO3 mg/l.	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.	125.9	8.68	8.65	2	13	4	2129	1289	5	12	0.050	0.007	0.034	0.032	0.28	0.130	0.055
FEB.	129.5	8.85	8.85	1	21	2	2143	1190	3	7	0.048	0.018	0.030	0.031	0.04	0.140	0.014
MAR.	110.5	8.98	8.98	N	25	2	2083	1385	3	6	0.030	0.020	0.020	0.030	0.04	0.120	0.008
APR.	122.4	8.92	8.92		25	2	2143	1198	3	4	0.031	0.006	0.019	0.030	0.07	0.094	0.009
MAY	149.9	9.25	9.24		24	5	1510	894	2	4	0.035	0.006	0.016	0.020	0.05	0.016	0.008
JUNE	142.0	9.14	9.12		21	1	1794	1038	2	5	0.025	0.008	0.012	0.014	0.03	0.010	0.007
JULY	90.8	9.38	9.33		13	3	2014	1098	4	7	0.026	0.010	0.016	0.016	0.05	0.023	0.009
AUG.	56.6	9.81	9.80		26	1	2216	1178	11	13	0.045	0.010	0.018	0.009	1.70	0.012	0.009
SEPT	52.6	9.59	9.59		22	1	2413	1304	4	6	0.048	0.056	0.023	0.016	0.09	0.011	0.013
OCT.	82.9	9.64	9.64		28	2	2335	1420	4	6	0.036	0.014	0.014	0.026	0.04	0.007	0.007
NOV.	75.2	9.54	9.53		27	3	2173	1242	6	9	0.029	0.003	0.018	0.029	0.03	0.013	0.004
DEC.	99.2	9.15	9.11		27	3	2839	1650	3	11	0.034	0.016	0.019	0.022	0.14	0.054	0.009
# OF SAMPLES	12	12	12	3	12	12	12	12	12	12	12	12	12	12	12	12	12
HIGH	149.9	9.81	9.80	2	28	5	2839	1650	11	13	0.050	0.056	0.034	0.032	1.70	0.140	0.055
LOW	52.6	8.68	8.65	N	13	1	1510	894	2	4	0.025	0.003	0.012	0.009	0.03	0.007	0.004
AVG	103.1	9.24	9.22	1	23	3	2149	1241	4	7	0.036	0.015	0.020	0.023	0.21	0.053	0.013

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : P-14 - PANEL - TAILINGS POND TREATED EFFLUENT

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U308 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
JAN.	125.9	5.200		4.85	11.50	3.78	6.60			0.11	0.100		
FEB.	129.5	4.300	320.7	3.75	12.52	3.57	4.53			0.16	0.030		
MAR.	110.5	6.000	453.0	5.90	12.00	3.90	4.85			0.18	0.020		
APR.	122.4	10.000	431.0	6.80	11.25	4.57	5.45			0.13	0.010		
MAY	149.9	7.100	306.0	4.42	9.36	2.54	3.40			0.07	0.020	21.30	
JUNE	142.0	8.600	333.0	3.43	9.30	3.46	4.44			0.05	0.010		
JULY	90.8	9.000	395.0	1.73	11.99	3.06	3.93			0.02	0.030		
AUG.	56.6	7.000	159.0	1.22	18.25	6.50	7.38			0.37	0.030		
SEPT	52.6	5.600	477.0	1.16	14.92	1.76	2.51			0.05	0.020		
OCT.	82.9	7.000	454.0	0.58	14.00	3.54	4.50			0.09	0.030		
NOV.	75.2	14.000	669.0	0.68	13.75	2.68	3.50			0.06	0.030		
DEC.	99.2	14.000	445.0	0.50	13.50	4.63	5.05			0.08	0.150		
# OF SAMPLES	12	12	11	12	12	12	12			12	12	1	
HIGH	149.9	14.000	669.0	6.80	18.25	6.50	7.38			0.37	0.150	21.30	
LOW	52.6	4.300	159.0	0.50	9.30	1.76	2.51			0.02	0.010	21.30	
AVG	103.1	8.150	403.9	2.92	12.70	3.66	4.68			0.11	0.040	21.30	

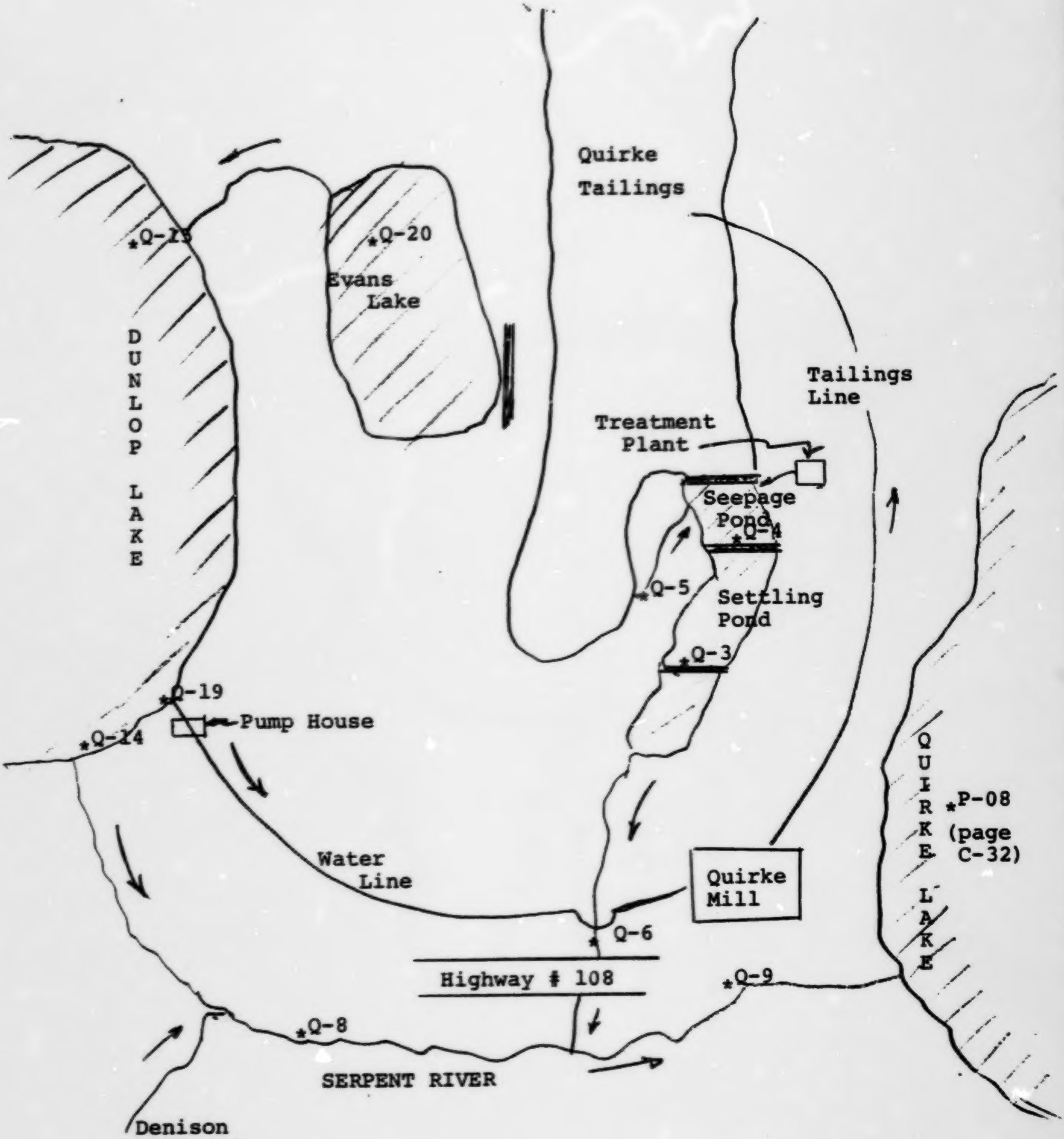
ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

APPENDIX C - IV

BASIC QUIRKE DATA

C - IV - 1 READINGS OF 1976 & 1983 FEED & DISCHARGE OF TREATMENT PLANT

C - IV - 2 SERPENT RIVER ABOVE & BELOW QUIRKE EFFLUENT STREAM


Quirke
Tailings
Q-15
Q-20
Evans
Lake
DUNLOP LAKE
Tailings
Line
Treatment
Plant
Seepage
Pond
Q-4
Q-5
Settling
Pond
Q-3
Q-19
Pump House
Q-14
QUIRKE LAKE
P-08
(page
C-32)
Water
Line
Quirke
Mill
Q-6
Highway # 108
Q-9
Q-8
SERPENT RIVER
Denison
Effluent

SCHEMATIC MAP - QUIRKE AREA SAMPLE POINTS

C - IV - 1 - QUIRKE

1976 & 1983 READINGS

- IN SPILLWAY, AHEAD OF BARIUM CHLORIDE ADDITION - Q - 05
- AT DAM D, BEFORE SECOND BARIUM CHLORIDE ADDITION - Q - 04
- QUIRKE TAILS POND EFFLUENT, AFTER BARIUM CHLORIDE ADDITIONS - Q - 03

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE
SAMPLE LOCATION: Q-5 QUIRKE - TAILS POND EFFLUENT, BEFORE BARIUM, IN SPILLWAY
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE MONTH	FLOW IGPM	PH	ACIDITY CACO3 MG/L	ALKALINITY CACO3 MG/L	S.S. MG/L	T.D.S. MG/L	SULPHATE SO4 MG/L	NITRATE NO3 AS N MG/L	AMMONIA NH3 AS N MG/L	RADIUM RA PC/L	URANIUM U308 MG/L	COPPER CU MG/L	IRON FE MG/L	COBALT CO MG/L	ZINC ZN MG/L	MANGANESE MN MG/L	LEAD PB MG/L	NICKE NI MG/L
JAN.	0	8.38	0	31	0	2487	1524	0.	0.	959.3	0.	0.	.22	0.	0.	0.	0.	0.
FEB.	0	8.60	0	37	0	2286	1280	0.	0.	817.8	0.	0.	.20	0.	0.	0.	0.	0.
MAR.	0	8.66	0	42	0	2347	1387	0.	0.	897.5	0.	0.	.10	0.	0.	0.	0.	0.
APR.	0	8.82	0	40	0	1918	1016	0.	0.	612.2	0.	0.	.04	0.	0.	0.	0.	0.
MAY	0	8.88	0	41	0	0	0	0.	0.	881.8	0.	0.	0.	0.	0.	0.	0.	0.
JUNE	0	8.68	0	36	0	2462	1382	0.	0.	878.5	0.	0.	.11	0.	0.	0.	0.	0.
JULY	0	8.77	0	38	0	2534	1452	0.	0.	944.0	0.	0.	.10	0.	0.	0.	0.	0.
AUG.	0	8.63	0	30	0	2519	1109	0.	0.	793.5	0.	0.	.10	0.	0.	0.	0.	0.
SEPT	0	8.72	0	34	0	2725	1530	0.	0.	768.0	0.	0.	.10	0.	0.	0.	0.	0.
OCT.	0	8.63	0	30	0	2626	1501	0.	0.	688.2	0.	0.	.13	0.	0.	0.	0.	0.
NOV.	0	8.56	0	31	0	2742	1580	0.	0.	732.0	0.	0.	.09	0.	0.	0.	0.	0.
DEC.	0	8.72	0	39	0	2740	1556	0.	0.	836.8	0.	0.	.07	0.	0.	0.	0.	0.
# OF SAMPLES	0	12	0	12	0	11	11	0	0	12	0	0	11	0	0	0	0	0
HIGH	0	8.88	0	42	0	2742	1580	0.	0.	959.3	0.	0.	.22	0.	0.	0.	0.	0.
LOW	0	8.38	0	30	0	1918	1016	0.	0.	612.2	0.	0.	.04	0.	0.	0.	0.	0.
AVERAGE	0	8.67	0	36	0	2489	1393	0.	0.	817.5	0.	0.	.11	0.	0.	0.	0.	0.

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION: Q-4 QUIRKE - TAILS POND EFFLUENT, BEFORE SECOND BARIUM, AT DAM D
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE	FLOW	PH	ACIDITY	ALKALINITY	S.S.	T.D.S.	SULPHATE	NITRATE	AMMONIA	RADIUM	URANIUM	COPPER	IRON	COBALT	ZINC	MANGANESE	LEAD	NICKE
MONTH	IGPM		CACO3 MG/L	CACO3 MG/L	MG/L	MG/L	SO4 MG/L	NO3 AS N MG/L	NH3 AS N MG/L	RA PC/L	U308 MG/L	CU MG/L	FE MG/L	CO MG/L	ZN MG/L	MN MG/L	PB MG/L	NI MG/L
JAN.	4196	6.89	0	21	12	0	0	0.	0.	26.1	0.	0.	0.	0.	0.	0.	0.	0.
FEB.	4151	7.41	0	28	11	0	0	0.	0.	17.4	0.	0.	0.	0.	0.	0.	0.	0.
MAR.	4334	7.34	0	29	10	0	0	0.	0.	22.0	0.	0.	0.	0.	0.	0.	0.	0.
APR.	5110	7.13	0	18	11	0	0	0.	0.	14.2	0.	0.	0.	0.	0.	0.	0.	0.
MAY	5131	8.19	0	29	10	0	0	0.	0.	61.1	0.	0.	0.	0.	0.	0.	0.	0.
JUNE	4370	7.91	0	25	10	0	0	0.	0.	15.2	0.	0.	0.1	0.	0.	0.	0.	0.
JULY	3835	8.05	0	27	11	0	0	0.	0.	20.2	0.	0.	0.	0.	0.	0.	0.	0.
AUG.	2982	7.43	0	20	11	0	0	0.	0.	7.4	0.	0.	0.	0.	0.	0.	0.	0.
SEPT	2839	7.04	0	15	12	0	0	0.	0.	53.1	0.	0.	0.	0.	0.	0.	0.	0.
OCT.	4150	7.29	0	17	12	0	0	0.	0.	81.3	0.	0.	0.	0.	0.	0.	0.	0.
NOV.	3851	6.91	0	21	13	0	0	0.	0.	13.0	0.	0.	0.	0.	0.	0.	0.	0.
DEC.	3835	7.55	0	29	14	0	0	0.	0.	27.3	0.	0.	0.	0.	0.	0.	0.	0.
# OF SAMPLES	12	12	0	12	12	0	0	0	0	12	0	0	0	0	0	0	0	0
HIGH	5131	8.19	0	29	14	0	0	0.	0.	81.3	0.	0.	0.	0.	0.	0.	0.	0.
LOW	2839	6.89	0	15	10	0	0	0.	0.	7.4	0.	0.	0.	0.	0.	0.	0.	0.
AVERAGE	4065	7.43	0	23	11	0	0	0.	0.	29.8	0.	0.	0.	0.	0.	0.	0.	0.

I WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION: Q-3 QUIRKE - TAILS POND EFFLUENT, AFTER BARIUM, AT DAM E

YEAR: 1976

TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE	FLOW	PH	ACIDITY	ALKALINITY	S.S.	T.D.S.	SULPHATE	NITRATE	AMMONIA	RADIUM	URANIUM	COPPER	IRON	COBALT	ZINC	MANGANESE	LEAD	NICKEL
MONTH	IGPM		CACO3 MG/L	CACO3 MG/L	MG/L	MG/L	SO4 MG/L	NO3 AS N MG/L	NH3 AS N MG/L	RA PC/L	U3O8 MG/L	CU MG/L	FE MG/L	CO MG/L	ZN MG/L	MN MG/L	PB MG/L	NI MG/L
JAN.	0	6.90	0	22	9	2696	1464	77.33	16.90	4.5	.32	.02	.54	.03	.02	.41	.05	.03
FEB.	0	7.27	0	27	8	2440	1353	75.90	19.78	3.8	.37	.02	.09	.03	.01	.44	.06	.04
MAR.	0	7.21	0	28	7	2540	1376	77.95	24.13	8.6	.32	.02	.07	.03	.03	.46	.05	.03
APR.	0	6.46	0	13	5	1700	997	66.20	24.20	3.8	.28	.03	.24	.03	.04	.28	.07	.04
MAY	0	7.53	3	26	3	2340	1341	71.55	21.63	9.9	.29	.02	.64	.03	.01	.26	.08	.03
JUNE	0	7.31	0	23	3	2484	1394	77.90	19.83	4.0	0.	.02	.04	.02	.07	.30	.06	.03
JULY	0	7.30	0	24	3	2606	1436	74.66	17.90	4.3	0.	.02	.05	.03	.11	.26	.06	.07
AUG.	0	7.13	0	19	3	2600	1453	74.47	16.30	3.5	.13	.02	.08	.04	.02	.36	.06	.04
SEPT	0	6.79	0	14	4	2732	1490	73.25	17.40	18.3	.11	.03	.24	.05	.03	.34	.06	.05
OCT.	0	7.15	0	12	3	2764	1506	76.48	17.96	4.1	.20	.02	.12	.03	.02	.23	.05	.03
NOV.	0	7.10	0	20	6	2800	1507	81.97	25.05	5.0	.31	.02	1.10	.02	.03	.36	.08	.02
DEC.	0	7.48	0	27	7	2880	1543	87.00	26.94	11.0	.15	.02	.07	.03	.02	.19	.06	.03
# OF SAMPLES	0	12	1	12	12	12	12	12	12	12	10	12	12	12	12	12	12	12
HIGH	0	7.53	3	28	9	2880	1543	87.0	26.9	18.3	.37	.03	1.10	.05	.11	.46	.08	.07
LOW	0	6.46	3	12	3	1700	997	66.2	16.3	3.5	.11	.02	.04	.02	.01	.19	.05	.02
AVERAGE	0	7.14	3	21	5	2548	1405	76.2	20.7	6.7	.25	.02	.27	.03	.03	.32	.06	.04

C-45

13: 4:43 19-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : Q-05 - QUIRKE - TAILS POND EFFLUENT, BEFORE BARIUM, IN SPILLWAY

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.	331.0	5.86	5.83	35	8	1	2380	1465		339					2.50		
FEB.	291.7	6.57	6.53	29	15	1	2359	1313		381					0.27		
MAR.	251.8	5.69	5.66	59	5	2	2326	1308	306	273					16.00		
APR.	297.7	5.04	4.94	45	4	3	1307	889		218					9.20		
MAY	306.0	3.71	3.70	106		3	1948	1246		242					0.11		
JUNE	261.8	3.98	3.97	66		1	2479	1336		227					3.90		
JULY	218.2	3.99	3.99	44		1	2641	1590		55					2.20		
AUG.	198.0	3.65	3.65	93		1	2810	1606		25					11.00		
SEPT	187.0	3.70	3.70	93		1	2774	1674		26					9.30		
OCT.	190.6	3.51	3.51	126		1	2811	1720		60					15.00		
NOV.	48.4	3.26	3.26	573		5	3767	2250		123					281.00		
DEC.	97.4	3.19	3.19	197		2	2931	1620		143					15.00		
# OF SAMPLES	12	12	12	12	4	12	12	12	1	12					12		
HIGH	331.0	6.57	6.53	573	15	5	3767	2250	306	381					281.00		
LOW	48.4	3.19	3.19	29	4	1	1307	889	306	25					0.11		
AVG	223.3	4.35	4.21	122	8	2	2544	1501	306	176					30.46		

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : Q-05 - QUIRKE - TAILS POND EFFLUENT, BEFORE BARIUM, IN SPILLWAY

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
JAN.	331.0				53.00	68.00							
FEB.	291.7				61.40	52.50							
MAR.	251.8				56.00	71.00							
APR.	297.7				38.00	49.00							
MAY	306.0				51.00	60.00							
JUNE	261.8				36.00	52.00							
JULY	218.2				61.00	52.00							
AUG.	198.0				62.00	50.00							
SEPT	187.0				62.00	47.00							
OCT.	190.6				68.00	47.00							
NOV.	48.4				103.00	46.00							
DEC.	97.4				97.00	60.00							
# OF SAMPLES	12				12	12							
HIGH	331.0				103.00	71.00							
LOW	48.4				36.00	46.00							
AVG	223.3				62.37	54.54							

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

13: 2:23 19-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : Q-04 - QUIRKE - TAILS POND EFFLUENT, BEFORE SECOND BARIUM, AT DAM D

FOR YEAR 1983
PAGE 1 OF 1 PAGE ONLY

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.	336.6	4.72	4.67	77		9			17								
FEB.	310.6	5.75	5.73	72	10	7			12	355							
MAR.	266.5	5.15	5.12	104	2	10			12	270							
APR.	352.3	4.45	4.40	108	1	12			10	151							
MAY	451.7	3.70	3.70	137		16			9	128							
JUNE	375.8	4.20	4.18	87		4			15	99							
JULY	248.9	3.91	3.91	101		3			2	14							
AUG.	208.6	3.63	3.63	163		2			2	8							
SEPT	217.4	3.67	3.67	157		2			1	8							
OCT.	222.4	3.47	3.47	184		2			2	15							
NOV.	72.1	3.31	3.31	592		12			2	75							
DEC.	117.8	3.22	3.22	295		10			2	7							
# OF SAMPLES	12	12	12	12	3	12			12	11							
HIGH	451.7	5.75	5.73	592	10	16			17	355							
LOW	72.1	3.22	3.22	72	1	2			1	7							
AVG	265.0	4.10	4.02	173	4	7			7	103							

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

- C-49 -

TABLE CONSISTS OF TOTAL LOADINGS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	ACIDITY as CaCO3 Kg/MON	ALKALINITY as CaCO3 Kg/MON	SUSPENDED SOLIDS Kg/MON	TOTAL DISSOLVED SOLIDS Kg/MON	SULPHATE SO4 Kg/MON	RADIUM DISSOLVED UC/MON	RADIUM TOTAL UC/MON	LEAD Pb Kg/MON	ZINC Zn Kg/MON	NICKEL Ni Kg/MON	COBALT Co Kg/MON	IRON Fe Kg/MON	MANGANESE Mn Kg/MON	COPPER Cu Kg/MON
JAN.	336.6	69674		8114			15100								
FEB.	310.6	53779	7363	4884			8828	266721							
MAR.	266.5	74485	1428	7138			8566	192729							
APR.	352.3	98383	776	10652			8765	137417							
MAY	451.7	165166		19357			10586	154249							
JUNE	375.8	84296		3653			14610	96427							
JULY	248.9	67363		1833			1333	9333							
AUG.	208.6	91161		1341			838	4469							
SEPT	217.4	88290		845			676	4696							
OCT.	222.4	109797		1192			894	8936							
NOV.	72.1	110567		2149			420	14016							
DEC.	117.8	93184		3049			505	2145							
HIGH	451.7	165166	7363	19357			15100	266721							
LOW	72.1	53779	776	845			420	2145							
AVG	265.0	92179	3189	5351			5927	81013							

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LDG1

12:58:23 19-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : D-03 - QUIRKE - TAILS POND EFFLUENT, AFTER BARIUM, AT DAM E

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.	336.6	8.85	8.83		60	3	2396	1466	10	24	0.003	0.005	0.033	0.014	1.40	0.200	0.018
FEB.	310.6	8.14	8.04		52	3	2204	1329	5	58	0.022	0.089	0.067	0.043	7.70	0.310	0.055
MAR.	267.0	7.97	7.88		42	3	2310	1247	1	16	0.033	0.016	0.040	0.044	0.38	0.280	0.056
APR.	352.3	7.95	7.87		34	4	1139	1059	1	12	0.025	0.023	0.048	0.074	6.50	0.550	0.025
MAY	451.7	7.22	6.99	104	23	5	2323	1418	2	37	0.025	0.024	0.084	0.069	4.50	0.610	0.030
JUNE	375.7	8.50	8.47		45	2	2524	1477	7	12	0.030	0.013	0.026	0.014	0.44	0.220	0.008
JULY	248.9	7.77	7.69	78	27	2	2678	1597	2	4	0.043	0.007	0.055	0.041	1.10	0.490	0.013
AUG.	208.6	7.70	7.54	175	35	2	2836	1622	1	2	0.017	0.026	0.057	0.063	5.50	0.750	0.013
SEPT	217.4	8.50	8.43		51	2	2588	1586	1	2	0.042	0.006	0.031	0.024	0.33	0.220	0.014
OCT.	222.4	8.49	8.46		42	3	3090	1678	1	3	0.047	0.012	0.048	0.050	1.40	0.680	0.018
NOV.	72.1	8.20	8.09		101	19	2733	1677	1	5	0.052	0.056	0.110	0.150	67.00	1.300	0.260
DEC.	117.8	9.56	9.55		186	119	2166	1632	1	5	0.110	0.170	0.150	0.150	46.00	1.500	0.077
# OF SAMPLES	12	12	12	3	12	12	12	12	12	12	12	12	12	12	12	12	12
HIGH	451.7	9.56	9.55	175	186	119	3090	1678	10	58	0.110	0.120	0.150	0.150	67.00	1.500	0.260
LOW	72.1	7.22	6.99	78	23	2	1139	1059	1	2	0.003	0.005	0.026	0.014	0.33	0.200	0.008
AVG	265.1	8.24	8.13	119	58	14	2416	1482	3	15	0.037	0.033	0.064	0.061	11.85	0.593	0.049

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : Q-03 - QUIRKE - TAILS POND EFFLUENT, AFTER BARIUM, AT DAM E

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
JAN.	336.6	15.000		1.08	50.75	57.75	60.55			0.23	0.017		
FEB.	310.6	16.000		1.22	54.80	60.63	63.30			0.30	0.025		
MAR.	267.0	15.600		1.56	55.00	67.16	69.36			0.30	0.010		
APR.	352.3	14.000		1.38	36.50	47.13	49.30			0.16	0.040		
MAY	451.7	14.000	416.0	0.94	40.50	55.45	57.45			0.21	0.010	13.00	
JUNE	375.7	9.800		1.17	37.60	48.26	49.98			0.22	0.010		
JULY	248.9	11.000		1.15	51.70	44.50	46.88			0.13	0.010		
AUG.	208.6	13.000		0.46	47.25	43.55	45.88			0.15	0.020		
SEPT	217.4	8.200		0.45	48.50	40.78	42.55			0.27	0.020		
OCT.	222.4	14.000		1.50	62.75	42.35	44.53			0.38	0.020		
NOV.	72.1	15.000		1.08	47.50	34.40	35.78			0.40	0.060		
DEC.	117.8	19.000		1.17	60.60	51.84	54.42			0.64	0.050		
# OF SAMPLES	12	12	1	12	12	12	12			12	12	1	
HIGH	451.7	19.000	416.0	1.56	62.75	67.16	69.36			0.64	0.060	13.00	
LOW	72.1	8.200	416.0	0.45	36.50	34.40	35.78			0.13	0.010	13.00	
AVG	265.1	13.717	416.0	1.10	49.45	49.44	51.66			0.28	0.024	13.00	

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

C - IV - 2 - QUIRKE

1976 & 1983 READINGS

- QUIRKE EFFLUENT TO SERPENT RIVER - Q - 06
- SERPENT RIVER ABOVE QUIRKE EFFLUENT ADDITION - Q - 08 (includes Denison effluent)
- SERPENT RIVER BELOW QUIRKE EFFLUENT ADDITION - Q - 09 (includes both Quirke & Denison)

WATER SAMPLE DETAILED REPORT-CONCENTRATIONS-YEAR TO DATE

SAMPLE LOCATION: O-6 QUIRKE - TAILS EFFLUENT TO SERPENT RIVER AT HIGHWAY 108
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE MONTH	FLOW IGPM	PH	ACIDITY CACO3 MG/L	ALKALINITY CACO3 MG/L	S.S. MG/L	T.D.S. MG/L	SULPHATE SO4 MG/L	NITRATE NO3 AS N MG/L	AMMONIA NH3 AS N MG/L	RADIUM RA PC/L	URANIUM U308 MG/L	COPPER CU MG/L	IRON FE MG/L	COBALT CO MG/L	ZINC ZN MG/L	MANGANESE MN MG/L	LEAD PB MG/L	NICKE NI MG/L
JAN.	0	6.20	0	11	0	2240	0	72.20	14.80	1.0	0.	0.	0.	0.	0.	0.	0.	0.
FEB.	0	6.70	0	15	0	2156	1176	71.10	18.20	6.3	0.	0.	0.	0.	0.	0.	0.	0.
MAR.	0	6.05	0	7	0	1512	0	52.60	17.40	6.5	0.	0.	0.	0.	0.	0.	0.	0.
APR.	0	6.35	0	5	0	844	0	32.00	9.00	6.0	0.	0.	0.	0.	0.	0.	0.	0.
MAY	0	6.80	0	10	0	1568	848	46.30	13.60	2.0	0.	0.	0.	0.	0.	0.	0.	0.
JUNE	0	6.70	0	17	0	2340	0	70.50	16.80	5.9	0.	0.	0.	0.	0.	0.	0.	0.
JULY	0	6.50	0	14	0	2448	0	73.40	16.80	4.7	0.	0.	0.	0.	0.	0.	0.	0.
AUG.	0	6.70	0	16	0	2504	1428	75.60	13.20	4.0	0.	0.	0.	0.	0.	0.	0.	0.
SEPT	0	6.60	0	11	0	2596	0	69.00	16.30	10.2	0.	0.	0.	0.	0.	0.	0.	0.
OCT.	0	6.50	0	11	0	2732	0	82.40	15.70	2.4	0.	0.	0.	8.	0.	0.	0.	0.
NOV.	0	6.40	0	8	0	2688	1470	81.30	17.20	5.3	0.	0.	0.	0.	0.	0.	0.	0.
DEC.	0	6.75	0	22	0	2648	0	88.00	25.00	9.7	0.	0.	0.	0.	0.	0.	0.	0.
# OF SAMPLES	0	12	0	12	0	12	4	12	12	12	0	0	0	0	0	0	0	0
HIGH	0	6.80	0	22	0	2732	1470	88.0	25.0	10.2	0.	0.	0.	0.	0.	0.	0.	0.
LOW	0	6.05	0	5	0	844	848	32.0	9.0	1.0	0.	0.	0.	0.	0.	0.	0.	0.
AVERAGE	0	6.52	0	12	0	2189	1230	67.9	[illegible].2	5.3	0.	0.	0.	0.	0.	0.	[illegible]	0.

I WATER SAMPLE DETAILED REPORT-CONCENTRATIONS-YEAR TO DATE

SAMPLE LOCATION: O-8 QUIRKE - SERPENT RIVER ABOVE EFFLUENT ADDITION AT MINE ROAD
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE	FLOW	PH	ACIDITY	ALKALINITY	S.S.	T.D.S.	SULPHATE	NITRATE NO3	AMMONIA NH3	RADIUM	URANIUM	COPPER	IRON	COBALT	ZINC	MANGANESE	LEAD	NICKE
			CACO3	CACO3			SO4	AS N	AS N	RA	U3O8	CU	FE	CO	ZN	MN	PB	NI
MONTH	IGPM		MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	PC/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L
JAN.	22400	7.00	0	9	0	224	0	5.40	4.20	1.9	0.	0.	0.	0.	0.	0.	0.	0.
FEB.	19500	7.15	0	9	0	276	106	6.50	4.00	1.0	0.	0.	0.	0.	0.	0.	0.	0.
MAR.	18000	7.86	0	14	0	276	0	12.90	7.00	1.0	0.	0.	0.	0.	0.	0.	0.	0.
APR.	52000	7.15	0	11	0	268	0	7.00	4.00	8.8	0.	0.	0.	0.	0.	0.	0.	0.
MAY	53200	7.00	5	7	0	148	69	3.70	2.40	4.0	0.	.03	.07	.03	.01	.04	.08	.03
JUNE	27200	7.35	0	8	0	120	0	8.10	2.60	3.3	0.	0.	0.	0.	0.	0.	0.	0.
JULY	10000	6.70	0	8	0	208	0	6.70	2.60	4.3	0.	0.	0.	0.	0.	0.	0.	0.
AUG.	5000	6.85	0	8	0	368	200	14.00	3.90	7.1	0.	0.	0.	0.	0.	0.	0.	0.
SEPT	500	6.95	0	9	0	1476	0	52.00	20.40	3.6	0.	0.	0.	0.	0.	0.	0.	0.
OCT.	200	7.10	0	16	0	2132	0	70.00	24.80	4.3	0.	0.	0.	0.	0.	0.	0.	0.
NOV,	1000	7.20	0	22	0	2200	1134	74.30	30.00	2.8	0.	0.	0.	0.	0.	0.	0.	0.
DEC.	1000	7.55	0	35	0	2572	0	90.00	40.00	3.4	0.	0.	0.	0.	0.	0.	0.	0.
# OF SAMPLES	12	12	1	12	0	12	4	12	12	12	0	1	1	1	1	1	1	1
HIGH	53200	7.86	5	35	0	2572	1134	90.0	40.0	8.8	0.	.03	.07	.03	.01	.04	.08	.03
LOW	200	6.70	5	7	0	120	69	3.7	2.4	1.0	0.	.03	.07	.03	.01	.04	.08	.03
AVERAGE	17500	7.15	5	13	0	855	377	29.2	12.2	3.8	0.	.03	.07	.03	.01	.04	.08	.03

WATER SAMPLE DETAILED REPORT-CONCENTRATIONS-YEAR TO DATE
SAMPLE LOCATION: Q-9 QUIRKE - SERPENT R. BELOW EFFLUENT ADDITION AT FLOW STATION
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE MONTH	FLOW IGPM	PH	ACIDITY CACO3 MG/L	ALKALINITY CACO3 MG/L	S.S. MG/L	T.D.S. MG/L	SULPHATE SO4 MG/L	NITRATE NO3 AS N MG/L	AMMONIA NH3 AS N MG/L	RADIUM RA PC/L	URANIUM U3O8 MG/L	COPPER CU MG/L	IRON FE MG/L	COBALT CO MG/L	ZINC ZN MG/L	MANGANESE MN MG/L	LEAD PB MG/L	NICKE NI MG/L
JAN.	33400	6.65	0	10	0	596	0	16.70	4.90	2.3	0.	0.	0.	0.	0.	0.	0.	0.
FEB.	0	6.70	0	12	0	628	352	10.60	6.40	3.5	0.	0.	0.	0.	0.	0.	0.	0.
MAR.	0	7.10	0	16	0	700	0	25.20	9.10	3.1	0.	0.	0.	0.	0.	0.	0.	0.
APR.	76000	0.	6	10	0	340	0	10.00	5.00	6.7	0.	0.	0.	0.	0.	0.	0.	0.
MAY	70602	6.90	5	7	0	251	142	7.70	4.00	11.2	0.	.03	.13	.03	.01	.07	.08	.03
JUNE	34000	6.95	0	10	0	500	0	17.20	4.20	4.2	0.	0.	0.	0.	0.	0.	0.	0.
JULY	15000	6.70	0	10	0	852	0	24.40	5.80	3.4	0.	0.	0.	0.	0.	0.	0.	0.
AUG.	7000	6.85	0	10	0	1088	432	33.90	5.20	3.2	0.	0.	0.	0.	0.	0.	0.	0.
SEPT	6000	6.95	0	10	0	1660	0	48.00	13.20	4.4	0.	0.	0.	0.	0.	0.	0.	0.
OCT.	8500	6.95	0	12	0	2000	0	62.60	16.30	4.1	0.	0.	0.	0.	0.	0.	0.	0.
NOV.	7000	7.10	0	13	0	2096	1082	61.50	17.80	3.8	0.	0.	0.	0.	0.	0.	0.	0.
DEC.	7000	6.95	0	23	0	2300	0	77.00	26.00	3.7	0.	0.	0.	0.	0.	0.	0.	0.
# OF SAMPLES	10	11	2	12	0	12	4	12	12	12	0	1	1	1	1	1	1	1
HIGH	76000	7.10	6	23	0	2300	1082	77.0	26.0	11.2	0.	.03	.13	.03	.01	.07	.08	.03
LOW	6000	6.65	5	7	0	251	142	7.7	4.0	2.3	0.	.03	.13	.03	.01	.07	.08	.03
AVERAGE	26450	6.89	5	11	0	1084	502	32.9	9.8	4.5	0.	.03	.13	.03	.01	.07	.08	.03

16:34:16 18-JAN-84

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : Q-06 - QUIRKE - TAILS EFFLUENT TO SERPENT RIVER AT HWY 108

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
JAN.					25.00	28.00							
FEB.					39.00	45.00							
MAR.					43.00	54.00							
APR.					23.00	28.00							
MAY					23.00	23.00							
JUNE					23.00	25.00							
JULY					38.80	42.50							
AUG.					54.00	42.00							
SEPT					47.50	36.00							
OCT.					35.00	33.00							
NOV.					42.00	26.00							
DEC.					22.00	14.00							
# OF SAMPLES					12	12							
HIGH					54.00	54.00							
LOW					22.00	14.00							
AVG					34.61	33.04							

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:34:15 13-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : Q-06 - QUIRKE - TAILS EFFLUENT TO SERPENT RIVER AT HWY 108

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.		6.70	6.70	16	10		1133		2	7							
FEB.		6.15	6.15	22	1		1623	996	1								
MAR.		9.10	9.10	N	87		1906	1118	1	3							
APR.		7.76	7.76	8	22		953		1								
MAY		5.85	5.85	22	5		1006	604	1	2							
JUNE		8.90	8.90	N	40		1303	779	11	12							
JULY		5.19	5.19	41	1		2156		1	2							
AUG.		6.40	6.40	16	5		2654	1572	1								
SEPT		8.06	8.06	4	32	4	2685		1	1							
OCT.		8.55	8.55	N	45	2	2421		1								
NOV.		8.70	8.70	N	38	3	1766	1036	1	1							
DEC.		6.60	6.60	8	13		812		2								
OF SAMPLES		12	12	12	12	3	12	6	12	7							
HIGH		9.10	9.10	41	87	4	2685	1572	11	12							
LOW		5.19	5.19	N	1	2	812	604	1	1							
AVG		7.33	7.22	11	25	3	1702	1018	2	4							

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:34:20 18-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : 0-08 - QUIRKE - SERPENT RIVER ABOVE EFFLUENT ADDITION AT MINE ROAD

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.	3076.0																
FEB.	2288.1	7.70	7.70	4	17		390		2	10							
MAR.	2467.0																
APR.	5831.5																
MAY	3596.6	6.25	6.25	4	5	1	318	171	4	5	0.002	0.010	0.004	0.008	0.08	0.078	0.007
JUNE	3371.5																
JULY	828.3																
AUG.		7.30	7.30	7	12	1	1080	577	2	3							
SEPT	392.0																
OCT.	2008.1	7.30	7.30	16	19	2	2457		2	5							
NOV.	2046.9	7.05	7.05	6	9	7	516	282	4	8							
DEC.	2520.3																
, OF SAMPLES	11	5	5	5	5	4	5	3	5	5	1	1	1	1	1	1	1
HIGH	5831.5	7.70	7.70	16	19	7	2457	577	4	10	0.002	0.010	0.004	0.008	0.08	0.078	0.007
LOW	392.0	6.25	6.25	4	5	1	318	171	2	3	0.002	0.010	0.004	0.008	0.08	0.078	0.007
AVG	2584.2	7.12	7.10	7	12	3	952	343	3	6	0.002	0.010	0.004	0.008	0.08	0.078	0.007

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:34:21 18-JAN-84

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : Q-08 - QUIRKE - SERPENT RIVER ABOVE EFFLUENT ADDITION AT MINE ROAD

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH .

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
FEB.	2288.1	3.000		0.44	12.00	8.60	10.30						
MAY	3596.6	4.000	64.0	0.12	7.20	5.80	6.40			0.01	0.050	13.00	
AUG.		13.000		2.30	25.00	17.70	18.40						
OCT.	2008.1	29.000		5.50	68.00	43.20	45.00						
NOV.	2046.9	5.100		1.70	13.00	9.50	10.50						
# OF SAMPLES	4	5	1	5	5	5	5			1	1	1	
HIGH	3596.6	29.000	64.0	5.50	68.00	43.20	45.00			0.01	0.050	13.00	
LOW	2008.1	3.000	64.0	0.12	7.20	5.80	6.40			0.01	0.050	13.00	
AVG	2484.9	10.820	64.0	2.01	25.04	16.96	18.12			0.01	0.050	13.00	

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

'3: 8:18 19-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : 0-09 - QUIRKE - SERPENT R. BELOW EFFLUENT ADDITION AT FLOW STATION

FOR YEAR 1983
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TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.	4885.0	7.70	7.66	6	10		357		5	17							
FEB.	3047.5	8.05	8.04	2	17		536	298	1	7							
MAR.	3584.0	7.95	7.94	N	24		792		1	5							
APR.	7820.0	7.15	7.12	4	20		621		2	13							
MAY	5710.0	6.35	6.35	4	5	1	355	218	2	5	0.008	0.010	0.009	0.013	0.20	0.140	0.005
JUNE	3917.2	7.35	7.35	5	10		334		3	4							
JULY	961.7	7.06	7.06	8	8		1025		2	3							
AUG.	687.3	6.95	6.95	8	10	12	1293	739	2	3							
SEPT	704.2	7.44	7.44	8	17		2100		1	1							
OCT.	3141.5	7.12	7.11	13	23	2	2144		1	2							
NOV.	2602.5	6.94	6.93	6	11	1	622	354	3	6							
DEC.	3350.0	6.70	6.70	4	7	1	259		2	3							
# OF SAMPLES	12	12	12	12	12	5	12	4	12	12	1	1	1	1	1	1	1
HIGH	7820.0	8.05	8.04	13	24	12	2144	739	5	17	0.008	0.010	0.009	0.013	0.20	0.140	0.005
LOW	687.3	6.35	6.35	N	5	1	259	218	1	1	0.008	0.010	0.009	0.013	0.20	0.140	0.005
AVG	3367.6	7.23	7.21	6	14	3	870	402	2	6	0.008	0.010	0.009	0.013	0.20	0.140	0.005

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : Q-09 - QUIRKE - SERPENT R. BELOW EFFLUENT ADDITION AT FLOW STATION

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
JAN.	4885.0	2.600		0.20	11.00	8.05	9.50						0.4
FEB.	3047.5	3.800		0.50	15.00	12.90	13.30						0.1
MAR.	3584.0	5.400		0.70	21.00	13.70	21.60						2.3
APR.	7820.0	6.000		0.40	16.00	14.60	15.30						4.3
MAY	5710.0	4.400	75.0	0.14	8.40	9.35	8.30			0.01	0.040	11.00	9.5
JUNE	3917.2	3.100		0.50	6.70	8.00	7.20						25.0
JULY	961.7	12.000		0.07	13.60	10.34	19.15						25.0
AUG.	687.3	12.000		1.00	29.00	23.50	20.70						22.4
SEPT	704.2	20.000		1.16	42.30	23.85	14.50						25.0
OCT.	3141.5	20.000		2.30	55.00	20.25	33.00						11.9
NOV.	2602.5	5.400		1.40	15.00	9.20	11.20						7.0
DEC.	3350.0	2.700		0.30	6.10	4.05	5.40						1.0
# OF SAMPLES	12	12	1	12	12	12	12			1	1	1	12
HIGH	7820.0	20.000	75.0	2.30	55.00	23.85	33.00			0.01	0.040	11.00	25.0
LOW	687.3	2.600	75.0	0.07	6.10	4.05	5.40			0.01	0.040	11.00	0.1
AVG	3367.6	8.117	75.0	0.72	19.92	13.15	14.93			0.01	0.040	11.00	11.2

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

APPENDIX C - V

NORDIC TREATMENT PLANT

C - V - 1 - FEED & DISCHARGE
1976 & 1983

C - V - 2 - FEED VOLUME & ASSAYS
by months
1975 to 1983 inclusive

APPENDIX C -V - 1

NORDIC TREATMENT PLANT

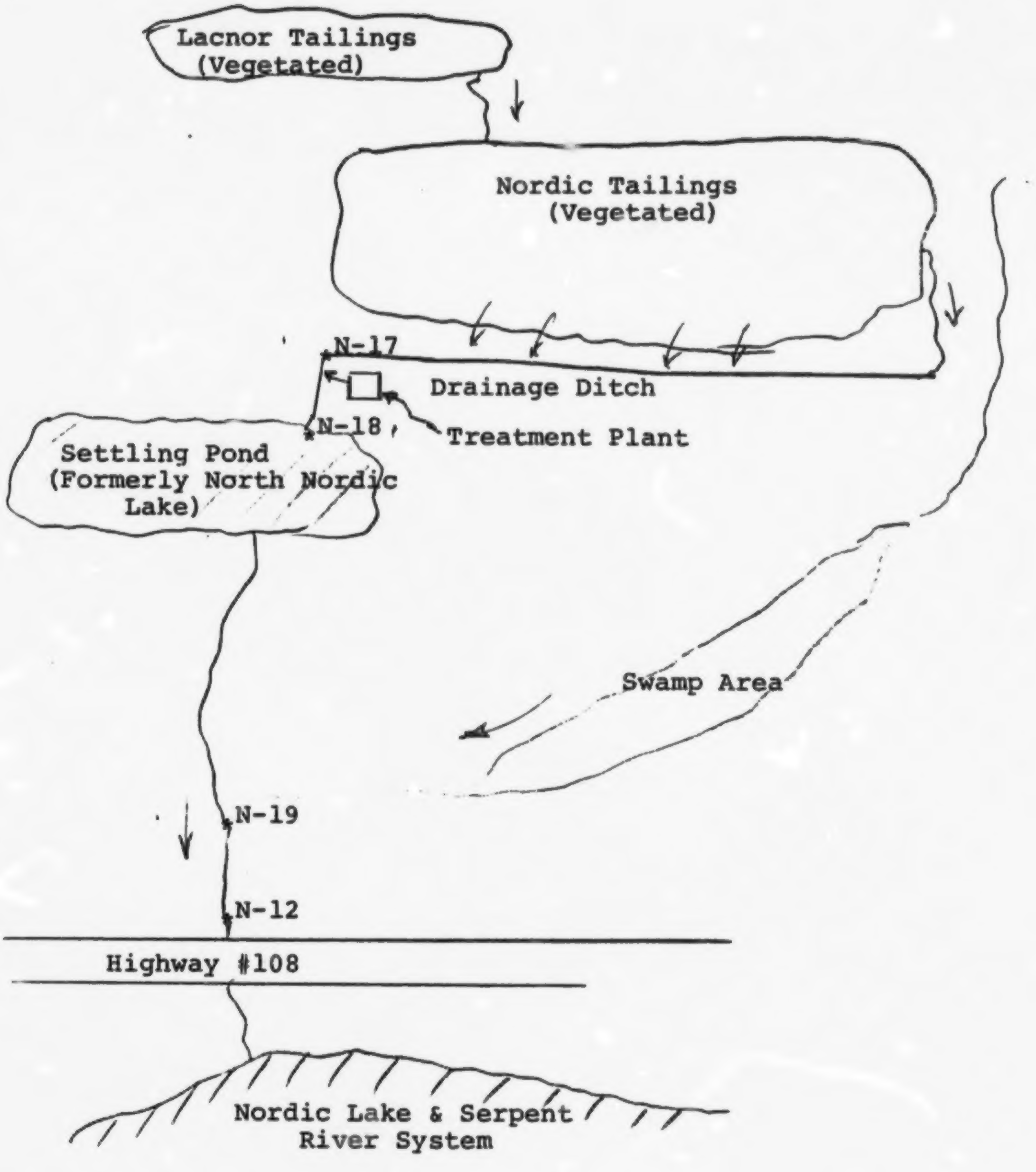
FEED & DISCHARGE
FROM TREATMENT PLANT
1976 & 1983

N - 12 - BUCKLES CREEK

N - 17 - FEED TREATMENT PLANT

N - 18 - TREATMENT PLANT DISCHARGE

N - 19 - NORTH NORDIC LAKE (SETTLING BASIN) OUTLET


Lacnor Tailings
(Vegetated)
Nordic Tailings
(Vegetated)
N-17
Drainage Ditch
N-18
Treatment Plant
Settling Pond
(Formerly North Nordic
Lake)
Swamp Area
N-19
N-12
Highway #108
Nordic Lake & Serpent
River System

SCHEMATIC MAP - NORDIC AREA SAMPLE POINTS

1 WATER SAMPLE DETAILED REPORT-CONCENTRATIONS-YEAR TO DATE

SAMPLE LOCATION: N-12 NORDIC BUCKLES CREEK AT HIGHWAY 108
YEAR: 1976

TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE MONTH	FLOW IGPM	PH	ACIDI- ITY CACO3 MG/L	ALKA- LINITY CACO3 MG/L	S.S. MG/L	T.D.S. MG/L	SUL- PHATE SO4 MG/L	NIT- RATE NO3 AS N MG/L	AMM- ONIA NH3 AS N MG/L	RADIUM RA PC/L	URAN- IUM U308 MG/L	COPPER CU MG/L	IRON FE MG/L	COBALT CO MG/L	ZINC ZN MG/L	MAN- GANESE MN MG/L	LEAD PB MG/L	NICKE NI MG/L
JAN.	2585	6.50	0	0	0	812	0	0.	0.	1.0	0.	0.	0.	0.	0.	0.	0.	0.
FEB.	1220	6.60	0	0	0	980	0	0.	0.	1.9	0.	0.	0.	0.	0.	0.	0.	0.
MAR.	2943	6.45	0	0	0	600	0	0.	0.	15.4	0.	0.	0.	0.	0.	0.	0.	0.
APR.	14387	5.70	0	0	0	212	0	0.	0.	11.1	0.	0.	0.	0.	0.	0.	0.	0.
MAY	3772	6.77	8	14	0	480	290	2.30	6.20	14.5	0.	.02	1.20	.03	.01	.26	.08	.03
JUNE	1040	7.20	0	0	0	1076	0	0.	0.	4.3	0.	0.	0.	0.	0.	0.	0.	0.
JULY	1328	6.78	0	0	0	948	0	0.	0.	7.5	0.	0.	0.	0.	0.	0.	0.	0.
AUG.	205	6.10	0	0	0	882	0	0.	0.	8.7	0.	0.	0.	0.	0.	0.	0.	0.
SEPT	443	6.63	0	0	0	1341	0	0.	0.	5.1	0.	0.	0.	0.	0.	0.	0.	0.
OCT.	491	6.72	0	0	0	1391	0	0.	0.	2.7	0.	0.	0.	0.	0.	0.	0.	0.
NOV.	529	7.17	0	0	0	1283	0	0.	0.	4.3	0.	0.	0.	0.	0.	0.	0.	0.
DEC.	400	6.85	0	0	0	1237	0	0.	0.	3.3	0.	0.	0.	0.	0.	0.	0.	0.
# OF SAMPLES	12	12	1	1	0	12	1	1	1	12	0	1	1	1	1	1	1	1
HIGH	14387	7.20	8	14	0	1391	290	2.3	6.2	15.4	0.	.02	1.20	.03	.01	.26	.08	.03
LOW	205	5.70	8	14	0	212	290	2.3	6.2	1.0	0.	.02	1.20	.03	.01	.26	.08	.03
AVERA	2445	6.62	8	14	0	936	290	2.3	6.2	6.6	0.	.02	1.20	.03	.01	.26	.08	.03

1 WATER SAMPLE DETAILED REPORT-CONCENTRATIONS-YEAR TO DATE

SAMPLE LOCATION: N-17 NORDIC - FEED TO NORDIC TREATMENT PLANT
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE MONTH	FLOW IGPM	PH	ACIDITY CACO3 MG/L	ALKALINITY CACO3 MG/L	S.S. MG/L	T.D.S. MG/L	SULPHATE SO4 MG/L	NITRATE NO3 AS N MG/L	AMMONIA NH3 AS N MG/L	RADIUM RA PC/L	URANIUM U308 MG/L	COPPER CU MG/L	IRON FE MG/L	COBALT CO MG/L	ZINC ZN MG/L	MANGANESE MN MG/L	LEAD PB MG/L	NICKE NI MG/L
JAN.	368	5.48	78	0	0	2043	0	0.	0.	8.4	0.	0.	0.	0.	0.	0.	0.	0.
FEB.	339	5.71	88	0	0	1816	0	0.	0.	11.8	0.	.02	29.00	.16	.08	3.50	.05	.10
MAR.	766	5.14	87	0	0	1776	0	0.	0.	11.9	0.	0.	0.	0.	0.	0.	0.	0.
APR.	2589	3.38	148	0	0	953	0	0.	0.	51.3	0.	0.	0.	0.	0.	0.	0.	0.
MAY	726	3.26	202	0	0	1440	944	6.50	10.80	25.3	0.	.08	85.00	.18	.10	3.00	.10	.17
JUNE	272	3.43	126	0	0	1889	0	0.	0.	20.4	0.	0.	0.	0.	0.	0.	0.	0.
JULY	162	3.17	151	0	0	2152	0	0.	0.	17.6	0.	0.	0.	0.	0.	0.	0.	0.
AUG.	106	3.14	169	0	0	2583	0	0.	0.	9.3	0.	.03	20.00	.18	.09	5.10	.08	.15
SEPT	170	4.05	104	0	0	1962	0	0.	0.	17.9	0.	0.	0.	0.	0.	0.	0.	0.
OCT.	158	5.42	53	0	0	1769	1200	0.	0.	5.7	0.	0.	0.	0.	0.	0.	0.	0.
NOV.	158	5.70	42	0	0	2018	0	0.	0.	9.7	0.	.01	17.00	.07	.04	3.40	.05	.07
DEC.	173	5.74	50	0	0	1727	0	0.	0.	9.5	0.	.02	15.00	.05	.05	2.00	.02	.04
# OF SAMPLES	12	12	12	0	0	12	2	1	1	12	0	5	5	5	5	5	5	5
HIGH	2589	5.74	202	0	0	2583	1200	6.5	10.8	51.3	0.	.08	85.00	.18	.10	5.10	.10	.17
LOW	106	3.14	42	0	0	953	944	6.5	10.8	5.7	0.	.01	15.00	.05	.04	2.00	.02	.04
AVERAGE	499	4.47	108	0	0	1844	1072	6.5	10.8	16.6	0.	.03	33.20	.13	.07	3.40	.06	.11

W A T E R S A M P L E D E T A I L E D R E P O R T - C O N C E N T R A T I O N S - Y E A R T O D A T E

SAMPLE LOCATION: N-18 NORDIC - EFFLUENT FROM NORDIC TREATMENT PLANT
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE MONTH	FLOW IGPM	PH	ACIDITY CACO3 MG/L	ALKALINITY CACO3 MG/L	S.S. MG/L	T.D.S. MG/L	SULPHATE SO4 MG/L	NITRATE NO3 AS N MG/L	AMMONIA NH3 AS N MG/L	RADIUM RA PC/L	URANIUM U308 MG/L	COPPER CU MG/L	IRON FE MG/L	COBALT CO MG/L	ZINC ZN MG/L	MANGANESE MN MG/L	LEAD PB MG/L	NICKE NI MG/L
JAN.	0	10.02	0	0	0	0	0	0.	0.	1.4	0.	0.	0.	0.	0.	0.	0.	0.
FEB.	0	10.74	0	0	0	0	0	0.	0.	3.0	0.	0.	0.	0.	0.	0.	0.	0.
MAR.	0	9.54	0	0	0	0	0	0.	0.	2.6	0.	0.	0.	0.	0.	0.	0.	0.
APR.	0	7.57	0	0	0	0	0	0.	0.	9.7	0.	0.	0.	0.	0.	0.	0.	0.
MAY	0	9.43	0	0	0	0	0	0.	0.	3.2	0.	0.	0.	0.	0.	0.	0.	0.
JUNE	0	10.51	0	0	0	0	0	0.	0.	4.2	0.	0.	0.	0.	0.	0.	0.	0.
JULY	0	11.01	0	0	0	0	0	0.	0.	3.0	0.	0.	0.	0.	0.	0.	0.	0.
AUG.	0	11.18	0	0	0	0	0	0.	0.	3.2	0.	0.	0.	0.	0.	0.	0.	0.
SEPT	0	10.64	0	0	0	0	0	0.	0.	4.0	0.	0.	0.	0.	0.	0.	0.	0.
OCT.	0	10.39	0	0	0	0	0	0.	0.	3.1	0.	0.	0.	0.	0.	0.	0.	0.
NOV.	0	10.27	0	0	0	0	0	0.	0.	3.8	0.	0.	0.	0.	0.	0.	0.	0.
DEC.	0	10.18	0	0	0	0	0	0.	0.	4.0	0.	0.	0.	0.	0.	0.	0.	0.
# OF SAMPLES	0	12	0	0	0	0	0	0	0	12	0	0	0	0	0	0	0	0
HIGH	0	11.18	0	0	0	0	0	0.	0.	9.7	0.	0.	0.	0.	0.	0.	0.	0.
LOW	0	7.57	0	0	0	0	0	0.	0.	1.4	0.	0.	0.	0.	0.	0.	0.	0.
AVERAGE	0	10.12	0	0	0	0	0	0.	0.	3.8	0.	0.	0.	0.	0.	0.	0.	0.

1 W A T E R S A M P L E D E T A I L E D R E P O R T - C O N C E N T R A T I O N S - Y E A R T O D A T E

SAMPLE LOCATION: N-19 NORDIC - NORTH NORDIC LAKE UTLET
YEAR: 1976
TABLE CONSISTS OF AVERAGES FOR EACH MONTH

SAMPLE DATE MONTH	FLOW IGPM	PH	ACIDITY CACO3 MG/L	ALKALINITY CACO3 MG/L	S.S. MG/L	T.D.S. MG/L	SULPHATE SO4 MG/L	NITRATE NO3 AS N MG/L	AMMONIA NH3 AS N MG/L	RADIUM RA PC/L	URANIUM U308 MG/L	COPPER CU MG/L	IRON FE MG/L	COBALT CO MG/L	ZINC ZN MG/L	MANGANESE MN MG/L	LEAD PB MG/L	NICKE NI MG/L
JAN.	0	6.65	0	22	0	1600	951	8.00	13.40	2.0	0.	.02	.07	.03	.01	.35	.05	.03
FEB.	0	7.30	0	27	0	1612	953	7.90	11.20	4.4	0.	.02	.16	.02	.01	.33	.05	.03
MAR.	0	7.25	0.	30	0	1536	0	8.10	14.00	6.0	0.	.02	.10	.03	.01	.36	.05	.03
APR.	0	4.90	0	2	0	472	414	1.50	4.00	23.7	0.	.02	3.90	.03	.02	.39	.06	.04
MAY	0	7.72	3	29	0	1297	868	5.60	3.10	7.8	0.	.02	.26	.03	.01	.36	.08	.03
JUNE	0	7.72	0	24	0	1349	851	5.40	10.80	2.0	0.	.02	.13	.02	.06	.27	.05	.03
JULY	0	7.82	0	26	0	1430	817	5.60	9.00	2.6	0.	.02	.08	.02	.01	.10	.04	.02
AUG.	0	6.92	0	10	0	682	426	4.70	1.50	3.3	0.	.02	1.00	.02	.03	.26	.04	.02
SEPT	200	7.71	0	33	0	1483	938	25.00	7.80	3.2	0.	.02	.07	.02	.01	.08	.04	.02
OCT.	0	8.16	0	32	0	1524	968	6.80	12.60	2.7	0.	.02	.14	.02	.01	.22	.07	.03
NOV.	327	8.40	0	33	0	1571	997	7.30	11.40	3.8	0.	.01	.06	.02	.01	.14	.05	.02
DEC.	294	8.07	0	41	0	1625	1083	7.00	10.00	2.4	0.	.02	.40	.01	.01	.20	.02	.01
# OF SAMPLES	3	12	1	12	0	12	11	12	12	12	0	12	12	12	12	12	12	12
HIGH	327	8.40	3	41	0	1625	1083	25.0	14.0	23.7	0.	.02	3.90	.03	.06	.39	.08	.04
LOW	200	4.90	3	2	0	472	414	1.5	1.5	2.0	0.	.01	.06	.01	.01	.08	.02	.01
AVERAGE	273	7.38	3	25	0	1348	842	7.7	9.1	5.3	0.	.02	.53	.02	.02	.25	.05	.03

16:31: 8 18-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : N-12 - NORDIC - BUCKLES CREEK AT HWY 108

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITH-METIC MEAN	PH GEOM-ETRIC MEAN	ACID-ITY as CaCO3 mg/L	ALKAL-INITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SUL-PHATE SO4 mg/L	RADIUM DISS-OLVED. pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MAN-GANESE Mn mg/L	COPPER Cu mg/L
JAN.		6.40	6.40				239			3							
FEB.		6.88	6.88				832			2							
MAR.		6.80	6.80				604			4							
APR.		6.80	6.80				463			11							
MAY		7.00	7.00	5	19	2	921	619	5	6	0.019	0.008	0.010	0.009	0.89	0.340	0.007
JUNE		6.10	6.10				604			14							
JULY		7.00	7.00				876			6							
AUG.		4.60	4.60			3	1652			6							
SEPT		5.90	5.90			3	1662			3							
OCT.		6.50	6.50			6	1266			3							
NOV.		6.57	6.57			7	940			3							
DEC.		6.40	6.40			24	789			3							
OF SAMPLES		12	12	1	1	6	12	1	1	12	1	1	1	1	1	1	1
HIGH		7.00	7.00	5	19	24	1662	619	5	14	0.019	0.008	0.010	0.009	0.89	0.340	0.007
LOW		4.60	4.60	5	19	2	239	619	5	2	0.019	0.008	0.010	0.009	0.89	0.340	0.007
AVG		6.41	6.38	5	19	8	904	619	5	5	0.019	0.008	0.010	0.009	0.89	0.340	0.007

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:31: 9 18-JAN-84

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : N-12 - NORDIC - BUCKLES CREEK AT HWY 108

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
MAY		29.000	196.0	0.20	2.40	4.00	5.50			0.01	0.010	17.20	
# OF SAMPLES		1	1	1	1	1	1			1	1	1	
HIGH		29.000	196.0	0.20	2.40	4.00	5.50			0.01	0.010	17.20	
LOW		29.000	196.0	0.20	2.40	4.00	5.50			0.01	0.010	17.20	
AVG		29.000	196.0	0.20	2.40	4.00	5.50			0.01	0.010	17.20	

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:31:22 18-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : N-17 - NORDIC - FEED TO NORDIC TREATMENT PLANT

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.	62.6	3.70	3.65	685			3151		6						320.67	10.400	
FEB.	44.5	4.83	4.83	808			3426		4						398.25	9.700	
MAR.	119.4	3.61	3.61	533			1939		9	8					257.50	0.120	
APR.	181.7	3.70	3.66	356			1320		10	33					168.00	6.260	
MAY	83.9	3.38	3.36	631	N	20	3157	2023	13	51	0.035	0.110	0.150	0.260	296.80	11.000	0.022
JUNE	65.2	4.11	4.10	1084			4214		8	18					521.33	14.000	
JULY	22.0	3.93	3.90	1083			4511		9	9					539.00	15.000	
AUG.	28.0	4.92	4.92	1100		23	4277		8	10					542.60	15.000	
SEPT	32.2	4.32	4.17	1089		21	3984			9					512.50	13.000	
OCT.	55.0	3.30	3.28	915		45	2198			12					418.75	6.200	
NOV.	69.7	3.48	3.42	877		29	3933			10					429.50	9.900	
DEC.	45.3	3.76	3.75	879		28				9					474.75	11.000	
: OF SAMPLES	12	12	12	12	1	6	11	1	8	10	1	1	1	1	12	12	1
HIGH	181.7	4.92	4.92	1100	N	45	4511	2023	13	51	0.035	0.110	0.150	0.260	542.60	15.000	0.022
LOW	22.0	3.30	3.28	356	N	20	1320	2023	4	8	0.035	0.110	0.150	0.260	168.00	0.120	0.022
AVG	67.4	3.92	3.86	837	N	28	3283	2023	8	17	0.035	0.110	0.150	0.260	406.64	10.132	0.022

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

16:31:23 18-JAN-84

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : N-17 - NORDIC - FEED TO NORDIC TREATMENT PLANT

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
JAN.	62.6				3.30	12.00							
FEB.	44.5				9.32	8.60							
MAR.	119.4				3.10	8.10							
APR.	181.7				2.60	3.80							
MAY	83.9	123.000	245.0	0.02	3.00	11.50	14.90			0.01	0.370	14.50	
JUNE	65.2				5.70	18.00							
JULY	22.0				8.00	15.00							
AUG.	28.0				6.90	19.00							
SEPT	32.2				14.00	25.00							
OCT.	55.0				11.00	5.40							
NOV.	69.7				10.00	14.00							
DEC.	45.3				8.90	16.00							
# OF SAMPLES	12	1	1	1	12	12	1			1	1	1	
HIGH	181.7	123.000	245.0	0.02	14.00	25.00	14.90			0.01	0.370	14.50	
LOW	22.0	123.000	245.0	0.02	2.60	3.80	14.90			0.01	0.370	14.50	
AVG	67.4	123.000	245.0	0.02	7.15	13.03	14.90			0.01	0.370	14.50	

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

12:46:46 19-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : N-18 - NORDIC - EFFLUENT FROM NORDIC TREATMENT PLANT

FOR YEAR 1983
PAGE 1 OF 1 PAGE ONLY

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITH-METIC MEAN	PH GEOM-ETRIC MEAN	ACID-ITY as CaCO3 mg/L	ALKAL-INITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SUL-PHATE SO4 mg/L	RADIUM DISS-OLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MAN-GANESE Mn mg/L	COPPER Cu mg/L
JAN.		9.50	9.47														
FEB.		9.14	9.00														
MAR.		8.46	8.29														
APR.		8.19	8.02														
MAY		8.21	8.05														
JUNE		9.12	9.04														
JULY		9.83	9.70														
AUG.		10.65	10.59														
SEPT		11.27	11.24														
OCT.		9.85	9.69														
NOV.		8.73	8.55														
DEC.		8.85	8.70														
# OF SAMPLES		12	12														
HIGH		11.27	11.24														
LOW		8.19	8.02														
AVG		9.32	9.15														

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

12:49: 3 19-JAN-84

WATER SAMPLE DETAILED REPORT - CONCENTRATIONS - YEAR TO DATE

SAMPLE LOCATION : N-19 - NORDIC - NORTH NORDIC LAKE EFFLUENT

FOR YEAR 1983
PAGE 1 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	PH ARITHMETIC MEAN	PH GEOMETRIC MEAN	ACIDITY as CaCO3 mg/L	ALKALINITY as CaCO3 mg/L	S.S. mg/L	T.D.S. mg/L	SULPHATE SO4 mg/L	RADIUM DISSOLVED pC/L	RADIUM TOTAL pC/L	LEAD Pb mg/L	ZINC Zn mg/L	NICKEL Ni mg/L	COBALT Co mg/L	IRON Fe mg/L	MANGANESE Mn mg/L	COPPER Cu mg/L
JAN.		6.94	6.84	4	25	1	1827	1219		3	0.003	0.004	0.017	0.016	0.19	0.260	0.008
FEB.		8.05	8.04	1	28	1	2152	1210		2	0.022	0.009	0.013	0.013	0.07	0.250	0.010
MAR.		7.33	7.26	N	29	3	1253	824		3	0.030	0.020	0.010	0.020	1.60	0.220	0.005
APR.		8.25	8.20	5	12	9	1102	751		10	0.020	0.032	0.014	0.017	2.90	0.480	0.006
MAY		7.77	7.76	4	27	1	1792	1277	3	4	0.008	0.006	0.006	0.005	0.12	0.340	0.002
JUNE		6.29	5.74	8	23	1	1921	1251		8	0.028	0.005	0.014	0.014	7.30	0.470	0.006
JULY		7.25	7.25	5	12	5	2058	1332		4	0.016	0.013	0.021	0.011	0.25	0.370	0.006
AUG.		6.92	6.92	5	12	3	2179	1289		5	0.015	0.005	0.008	0.011	0.05	0.520	0.005
SEPT		7.05	7.04	5	13	2	2277	1457		3	0.032	0.013	0.023	0.013	0.15	0.750	0.009
OCT.		6.77	6.77	6	13	5	2204	1429		4	0.034	0.017	0.031	0.019	0.65	0.670	0.011
NOV.		6.90	6.88	6	18	3	2186	1450		3	0.039	0.006	0.023	0.022	0.12	0.540	0.010
DEC.		7.00	7.00	6	15	3	2172	1429		3	0.027	0.017	0.019	0.022	0.06	0.570	0.004
# OF SAMPLES		12	12	12	12	12	12	12	1	12	12	12	12	12	12	12	12
HIGH		8.25	8.20	8	29	9	2277	1457	3	10	0.039	0.032	0.031	0.022	7.30	0.750	0.011
LOW		6.29	5.74	N	12	1	1102	751	3	2	0.003	0.004	0.006	0.005	0.05	0.220	0.002
AVG		7.21	7.11	5	19	3	1927	1243	3	4	0.023	0.012	0.017	0.015	1.12	0.453	0.007

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

WATER SAMPLE DETAILED REPORT CONCENTRATIONS
YEAR TO DATE

SAMPLE LOCATION : N-19 - NORDIC - NORTH NORDIC LAKE EFFLUENT

FOR YEAR 1983
PAGE 2 OF 2 PAGES

TABLE CONSISTS OF AVERAGE CONCENTRATIONS FOR EACH MONTH

SAMPLE DATE MONTH	FLOW L/s	MAGNESIUM Mg mg/L	CALCIUM Ca mg/L	NITRITE NO2 as N mg/L	NITRATE NO3 as N mg/L	AMMONIA NH3 as N mg/L	KJELDAHL N as N mg/L	OIL AND GREASE mg/L	TURBIDITY JACKSON UNITS	URANIUM U3O8 mg/L	PHOSPHORUS P mg/L	CHLORIDE Cl mg/L	TEMPERATURE CEL
JAN.				0.40	4.70	9.80	12.60						
FEB.				0.63	5.26	6.10	6.73						
MAR.		97.000		0.10	2.60	6.70	8.70						
APR.				0.06	2.60	5.60	7.80						
MAY		55.000	395.0	0.20	4.40	8.70	12.10			0.04	0.010	18.30	
JUNE				0.25	3.80	9.00	11.50						
JULY				0.06	5.50	7.80	10.00						
AUG.				0.09	5.50	9.50	12.00						
SEPT				0.16	5.00	9.50	11.50						
OCT.				0.18	5.90	2.80	4.80						
NOV.				0.30	5.30	9.80	11.80						
DEC.				0.30	4.90	10.60	12.30						
# OF SAMPLES		2	1	12	12	12	12			1	1	1	
HIGH		97.000	395.0	0.63	5.90	10.60	12.60			0.04	0.010	18.30	
LOW		55.000	395.0	0.06	2.60	2.80	4.80			0.04	0.010	18.30	
AVG		76.000	395.0	0.23	4.62	7.99	10.15			0.04	0.010	18.30	

ANALYTICAL INDICATORS : N - NOT DETECTED BLANK - NO DATA AVAILABLE

APPENDIX C - V - 2

NORDIC TREATMENT PLANT

FEED VOLUME & ASSAYS

by months

1975 to 1983 INCLUSIVE

N - 17

Note: The acidity level, and the level of TDS and Iron given all refer to the feed solution entering the flume of the treatment area, before any lime is added to neutralize the solution.

DATE	Volume	Acidity	Lime Added		T.D.S.	Fe
1975	1000 m^3/mo	$CaCO_3$ mg/L	t/mo	mg/L	mg/L	mg/L
Jan	75.4	98	28	371	-	-
Feb	50.3	60	26.6	529	-	-
Mar	55.5	52	28	505	-	-
Apr	601.1	102	64.5	107	-	-
May	179.8	143	54.6	304	-	-
June	261.5	102	42.7	163	1634	-
July	41.3	75	26.6	644	1665	-
Aug	42.7	81	22.1	518	2214	3
Sept	49.0	85	33.7	688	1744	-
Oct	42.2	78	26	616	1897	-
Nov	140.3	175	35.8	255	1679	43
Dec	170.3	135	40.8	240	2056	23
Total	1709.4		429.4			
Average		98.8		403.3	1698.4	23
1976						
Jan	74.7	78	33.8	452	2043	-
Feb	62.1	88	30.6	493	1816	29
Mar	155.4	87	33.7	217	1776	-
Apr	508.4	148	98	193	953	-
May	147.3	202	47.5	322	1440	85
June	53.4	126	32.1	601	1889	-
July	32.9	151	26	790	2152	-
Aug	21.5	169	18.3	851	2583	20
Sept	33.4	104	27.6	826	1962	-
Oct	32	53	21.6	675	1769	-
Nov	31	42	24.5	790	2018	17
Dec	35	50	27.6	786	1727	15
Total	1187.2		421.3			
Average		108.1		583	1844	27.67

DATE	Volume	Acidity	Lime Added		T.D.S.	Fe
1977	1000 m^3/mo	$CaCO_3$ mg/L	t/mo	mg/L	mg/L	mg/L
Jan	54	50	18.3	341	1748	-
Feb	54.9	46	18.3	334	1832	23
Mar	327.3	146	50.5	155	1385	-
Apr	325.0	133	85.7	264	1002	-
May	88.1	87	50.5	574	1824	23
June	33.1	121	21.4	647	1577	-
July	5.7	118	18.3	3230	2116	-
Aug	19.1	117	16.9	886	2338	4
Sept	49.9	127	19.9	398	1912	-
Oct	128.5	262	45.9	358	1928	-
Nov	351.8	267	88.7	252	1400	55
Dec	84.4	243	41.4	490	2272	-
Total	1521.8		475.8			
Average		143		660	1777	26
1978						
Jan	40.6	217	37.7	931	2533	-
Feb	39.9	191	27.8	696	2566	60
Mar	49.2	200	37.3	758	2710	71
Apr	252.5	272	100	396	1248	64
May	200.8	292	81.9	408	998	50
June	44.7	258	39.2	877	2490	36
July	38.9	294	29.4	756	2728	41
Aug	33.7	316	43.1	1280	3018	27
Sept	67.6	335	64.7	1000	2237	-
Oct	239.7	326	103.9	434	2070	-
Nov	74	300	45.1	610	2515	85
Dec	58	313	45.1	778	2450	90
Total	1139.6		655.2			
Average		276.1		743.67	2213.58	58.2

DATE	Volume	Acidity	Lime Added		T.D.S.	Fe
1979	1000 m^3/mo	$CaCO_3$ mg/L	t/mo	mg/L	mg/L	mg/L
Jan	45.5	322	36.3	798	2329	97
Feb	60	328	41.2	687	2496	133
Mar	265.3	305	83.1	313	1567	84
Apr	508.3	210	137.2	270	702	1
May	182.6	305	125.4	687	1512	54
June	198.1	328	90.2	456	266	45
July	127.7	269	89.8	704	2166	103
Aug	191.9	291	101.6	530	2082	91
Sept	145.5	325	72.5	499	2073	98
Oct	256.1	300	76.4	298	2169	98
Nov	328.5	321	101.9	310	1702	96
Dec	135	336	92.6	686	1943	101
Total	2444.5		1048.2			
Average		303.3		519.8	1750.5	83.4
1980						
Jan	138.8	333	81.3	586	1709	99
Feb	92.7	400	47	507	2895	112
Mar	182.1	413	49	269	2814	155
Apr	500.2	291	135.2	270	662	72
May	136.4	461	84.1	617	1723	59
June	95.3	269	51	535	1844	104
July	70.2	328	53	755	2476	106
Aug	65.1	365	43	661	2205	128
Sept	76.7	384	55	717	2519	176
Oct	109.8	431	106	965	1787	134
Nov	78.6	464	74.5	948	2282	161
Dec	45.3	418	51	1127	2517	-
Total	1591.2		830.1			
Average		379.7		663	2036	118.7

DATE	Volume	Acidity	Lime Added		T.D.S.	Fe
1981	1000 m^3/mo	$CaCO_3$ mg/L	t/mo	mg/L	mg/L	mg/L
Jan	65.8	445	49	745	2552	190
Feb	96.1	382	43	447	2614	205
Mar	107.7	434	64.7	379	3070	224
Apr	398.1	355	90.2	227	2171	170
May	131.5	444	51	388	2477	165
June	181.2	412	64.7	357	2381	175
July	109	409	80.4	738	2567	176
Aug	78.2	509	35.3	451	2566	167
Sept	59.3	512	43.1	727	2330	153
Oct	150.2	428	98	653	1283	124
Nov	107	406	76.4	714	2505	225
Dec	105.3	512	70.6	670	2952	244
Total	1589.4		766.4			
Average		437.3		541.3	2455.6	176.5
1982						
Jan	117.9	504	54.9	466	--	-
Feb	95.8	515	51	532	3038	230
Mar	107.4	540	62.7	584	2811	262
Apr	520	328	168.6	324	931	119
May	184.8	510	95.1	515	2451	192
June	167.2	724	51	305	3401	330
July	57.6	706	37.2	646	3193	309
Aug	114.6	836	49	427	3891	382
Sept	192.6	620	96	499	2089	245
Oct	272.7	645	131.3	482	2520	251
Nov	206.8	539	105.8	511	1951	245
Dec	290.9	565	129.4	445	1806	215
Total	2328.3		1032			
Average		586		478	2549.6	252.7

DATE	Volume	Acidity	Lime	Added	T.D.S.	Fe
1983	1000 m^3/mo	$CaCO_3$ mg/L	t/mo	mg/L	mg/L	mg/L
Jan	168	685	96.0	573	3151	320
Feb	108	808	54.9	510	3426	398
Mar	320	533	134.2	420	1939	257
Apr	471	356	147	312	1320	168
May	225	631	96	427	3157	297
June	169	1084	86.2	510	4214	521
July	59	1083	51	865	4511	539
Aug	75	1100	51	680	4277	543
Sept	83	1089	56.8	680	3584	513
Oct	147	915	84.3	572	2198	419
Nov	181	877	84.3	467	3932	430
Dec	117	879	66.6	567	--	475
Total	2123		1008.3			
Average		836.6		548.58	3246.27	406.67

APPENDIX C - VI

COMPARISONS OF MAY MONTH
STREAM LOADINGS 1976 - 1983

SAMPLE SPOTS

N-17 - Feed to Nordic Treatment Plant
PR-2 - Feed to Pronto Treatment Plant
Q-03 - Effluent Release to Serpent River
Quirke Treatment Plant

NORDIC - FEED TO TREATMENT PLANT SAMPLE POINT N - 17

			Stream Loading Tonnes/Month					
May of	pH	Vol of Sol'n 1000 m^3/mo	$CaCO_3$ Acidity	TDS	SO_4	Fe	Mn	Kjeldahl N
1983	3.38	224.7	141.7	709.1	454.4	66.7	2.5	3.3
1982	3.52	184.8	94.2	453.0	298.1	35.4	1.4	2.2
1981	3.95	127.2	58.3	325.4	199.2	21.7	1.1	2.2
1980	3.30	132.0	63.0	235.1	127.2	8.1	-	1.8
1979	3.30	182.8	57.5	285.0	185.3	10.2	0.9	2.0
1978	3.16	197.8	63.5	336.0	285.0	16.8	0.8	2.3
1977	4.39	85.2	7.7	168.0	109.0	2.1	0.4	1.4
1976	3.26	143.0	34.5	327.0	213.0	19.0	0.7	2.5

PRONTO - FEED TO TREATMENT PLANT - SAMPLE POINT PR - 2

			Stream Loading Tonnes/Month			
May of	pH	Vol of Sol'n 1000 m^3/mo	$CaCO_3$ acidity	TDS	SO_4	Fe
1983	3.36	145	14.7	98.9	66.1	3.9
1982	3.49	307	30.7	147.2	72.1	4.3
1981	3.30	247	32.2	159.2	105.2	5.9
1980	3.30	279	36.7	181.5	95.2	1.2
1979	3.40	273	49.5	198.0	143.8	6.5
1978	3.12	275	49.9	131.6	72.6	6.4
1977	2.95	162	34.9	249	190	10.4
1976	2.94	125	26.3	91	63	6.2

QUIRKE - TAILINGS POND EFFLUENT (after Ba & CaO) SAMPLE SPOT Q-03

			Stream Loading Tonnes/Month					
May of	pH	Sol'n kt/mo	$CaCo_3$ Alkalinity	TDS	SO_4	Fe	Mn	Kjeldahl N
1983	7.22	1210	27.8	2810	1715	5.4	0.7	70
1982	7.80	1021	22.7	2540	1725	1.2	0.2	80
1981	8.00	692	22.2	2187	1164	0.6	0.2	48
1980	8.70	1015	93	2579	1485	0.06	0.13	43
1979	8.90	1170	67	2534	1406	1.3	.05	35
1978	8.74	833	46	1794	1043	0.6	0.15	15
1977	8.40	863	35	2123	1171	0.3	0.2	22
1976	7.53	1007	26	2356	1350	0.6	0.26	22

APPENDIX C - VII

MINE WATER SAMPLES

C-VII-1 - DENISON - 1982 - AVERAGES
C-VII-2 - PANEL - YEARLY AVERAGES 1980-83
C-VII-3 & 4 - QUIRKE - YEARLY AVERAGES 1978-83

APPENDIX C - VII - 1

EXCERPT FROM DENISON REPORT
1982 MINE WATER VALUES

3.2.3 Minewater Recycling

The minewater for 1982 was pumped to the grinding plant at an average rate of 63.7 L/s (841 Igpm). A total of approximately 2 002 000 tonnes of water was treated by the lime neutralization facility.

All the minewater was fed into the grinding circuit and used as process water. Shown below is a chemical and radiological analysis of the minewater:

Element		
Iron as Fe	104	mg/L
Zinc as Zn	8.8	mg/L
Copper as Cu	2.3	mg/L
Cobalt as Co	0.1	mg/L
Nickel as Ni	1.1	mg/L
Manganese as Mn	16.4	mg/L
Lead as Pb	1.1	mg/L
Sulphate as SO4	1 165	mg/L
Calcium as Ca	194	mg/L
Chloride as Cl	345	mg/L
Ammonia as N	65	mg/L
Nitrate as N	81	mg/L
Nitrite as N	less 1	mg/L
Potassium as K	15.6	mg/L
Magnesium as Mg	14.8	mg/L
Uranium as U_3O_8	147	mg/L
Radium as Ra^{226} dissolved	275	pCi/L
Radium as Ra^{226} total	283	pCi/L
pH	2.8	

APPENDIX C - VII - 2

PANEL MINE WATER BEFORE TREATMENT-AVERAGES FOR YEAR PW - 01

Year	pH	Sol'n kt/m	$CaCO_3$ acidity mg/L	TDS mg/L	SO_4 mg/L	Fe mg/L	Cl mg/L	Mn mg/L	NO_3/N mg/L	NH_3/N mg/L	Ra pc/L	U_3O_8 mg/L
1983	3.30	30.3	407	1897	824	66	222	1.7	94	75	108	61
			(10)	(3)	(3)	(3)	(11)	(3)	(11)	(11)	(3)	
1982	2.65	36.2	-	1523	850	139	304	1.5	113	76	71	58
				(1)	(1)	(2)	(11)	(1)			(1)	
1981	3.57	27.9	1075	3778	2368	117	-	2.5	103	77	121	32
			(9)	(10)	(2)	(10)		(10)	(10)	(10)	(10)	(11)
1980	3.75	59.1	884	2938	1313	-	-	-	104	79	127	53
				(11)	(11)						(1)	

No earlier data.

** Note: Averages are from 12 readings/year unless noted ( )

APPENDIX C - VII - 3

QUIRKE 1 MINE WATER BEFORE TREATMENT - AVERAGE FOR YEAR QW - 01

Year	pH	Sol'n kt/m	$CaCO_3$ acidity mg/L	TDS mg/L	SO_4 mg/L	Fe mg/L	Cl mg/L	Mn mg/L	NO_3/N mg/L	NH_3/N mg/L	Ra pc/L	U_3O_8 mg/L
1983	2.8	38.3	439	783	485	40	9	0.7	3	2	45	16
				(4)	(4)	(3)		(4)			(4)	
1982	2.81	37.3	413	827	416	44	11	0.9	7	6	55	19
				(4)	(1)	(5)		(4)			(4)	
1981	2.82	35.6	454	1076	538	51	-	4.6	5	4	69	-
					(9)			(9)			(9)	
1980	3.0	33.4	539	1083	756	84	-	1.4	12	6	47	-
			(11)	(11)	(9)	(11)		(10)			(10)	
1979	3.0	-	441	935	625	77		1.6	16	11	78	-
1978	3.0	-	777	1672	1062	136		2.3	34	20	96	-
			(11)	(11)	(11)	(11)		(11)	(11)	(11)	(11)	

No earlier data.

** Note: Averages are from 12 readings/year unless noted ( ).

APPENDIX C - VII - 4

QUIRKE 2 MINE WATER BEFORE TREATMENT - AVERAGE FOR YEAR QW - 02

Year	pH	Sol'n kt/m	$CaCO_3$ acidity mg/L	TDS mg/L	SO_4 mg/L	Fe mg/L	Cl mg/L	MN mg/L	NO_3/N mg/L	NH_3/N mg/L	Ra pc/L	U_3O_8 mg/L
1983	2.81	87.1	638	1270	742	86	32	1.2	49	32	89	60
				(4)	(4)	(4)		(4)			(4)	
1982	2.87	97.5	601	1421	790	97	28	1.4	69	48	193	69
				(5)	(1)	(5)		(4)			(4)	
1981	2.89	90.1	586	1268	756	88	-	1.3	58	44	285	-
					(10)			(11)			(11)	
1980	3.00	98.8	590	1233	786	94		1.3	59	44	335	-
			(11)	(11)	(11)	(11)		(11)			(11)	
1979	3.20	-	446	967	555	71	-	1.0	56	43	437	-
			(11)	(11)	(11)	(11)		(11)	(11)	(11)	(11)	
1978	3.18	-	543	1151	633	81	-	1.1	63	46	552	

No earlier data.

** Note: Averages are from 12 readings/year unless noted ( ).

APPENDIX C - VIII

RIO ALGOM PLANTS - SUMMARY OF OPERATIONS

1982 & 1983

1982 EFFLUENT TREATMENT PLANT SUMMARY FOR PRONTO

	ITEM	JAN.	FEB.	MAR.	APR.	MAY	JUN.	JUL.	AUG.	SEP.	OCT.	NOV.	DEC.
EFFLUENT FLOW RATE	Days Plant Operated This Month:	PLANT SHUTDOWN	PLANT SHUTDOWN	6	30	31	24	PLANT SHUTDOWN	PLANT SHUTDOWN	8	31	30	22
	1) Maximum: Litres/sec. (IGPM)			215.2 (2840)	215.2 (2840)	135.6 (1790)	84.1 (1110)			208.0 (2745)	208.0 (2745)	148.5 (1960)	145.9 (1926)
	2) Minimum: Litres/sec. (IGPM)			35.0 (462)	84.2 (1111)	89.4 (1180)	19.1 (252)			84.0 (1109)	70.1 (925)	89.4 (1180)	35.0 (462)
	3) Average: Litres/sec. (IGPM)			189.5 (2500)	151.6 (2001)	114.5 (1511)	71.0 (937)			163.7 (2161)	107.3 (1416)	113.9 (1503)	114.2 (1509)
	Total Volume Treated: Litres x 10^6 (Imp. Ga. x 10^6)			98.237 (21.609)	392.947 (86.436)	306.677 (67.451)	147.226 (32.385)			113.149 (24.889)	287.392 (63.217)	259.229 (64.941)	217.071 (47.749)
LIME	pH: 1) Feed PR-2	2.89	2.91	3.60	5.08	3.49	3.05	2.95	2.85	3.00	3.08	3.13	3.51
	2) Discharge PR-4				7.05	9.80	9.28			7.75	8.99	9.75	5.98
	Acidity, mg/L as $CaCO_3$ 1) Feed PR-2	221	228	145	29	100	173	318	231	172	183	158	140
	Lime Consumption 1) Tonnes CaO (Metric) (Short Imp. Tons CaO)			7.8 (8.6)	45.1 (49.7)	29.4 (32.4)	21.6 (23.8)			9.8 (10.8)	35.3 (38.9)	43.1 (47.5)	29.4 (32.4)
	2) mg CaO/litre (lbs. CaO/1,000 Imp.)			79.4 (0.80)	114.8 (1.15)	95.9 (0.961)	146.7 (1.47)			86.6 (0.87)	122.8 (1.23)	146.0 (1.46)	135.4 (1.36)
BARIUM CHLORIDE	Dissolved Radium, pc/Litre 1) Feed PR-2	14.0	13.0	19.0	1.8	4.2	10.8	17.0	N.A.	11.0	11.0	12.0	7.0
	2) Discharge PR-4				1.5	2.2	2.1			2.0	2.5	3.0	4.0
	Barium Chloride Consumption 1) Kilograms (pounds)			500 (1100)	1150 (2530)	900 (1980)	650 (1430)			300 (660)	700 (1540)	800 (1760)	400 (880)
	2) mg/l (lbs./1,000 Imp. Gal)			5.09 (0.051)	2.93 (0.029)	2.93 (0.029)	4.41 (0.044)			2.65 (0.027)	2.44 (0.024)	2.71 (0.027)	1.84 (0.018)
Effluent at Property Boundary: PR-1	1) pH	6.37	6.64	6.73	7.23	7.14	6.87	7.28	7.10	6.64	6.69	7.11	6.73
	2) Total Radium pc/Litre	3.6	2.9	4.9	0.90	2.9	2.0	1.0	N.A.	3.0	3.0	3.0	2.0
Rainfall:	1) millimetres (inches)												

C-62

1982 EFFLUENT TREATMENT PLANT SUMMARY FOR NORDIC

	ITEM	JAN.	FEB.	MAR.	APR.	MAY	JUN.	JUL.	AUG.	SEP.	OCT.	NOV.	DEC.
EFFLUENT FLOW RATE	Days Plant Operated This Month:	31	28	31	30	31	30	31	31	30	31	30	31
	1) Maximum: Litres/sec. (IGPM)	59.5 (785)	55.7 (735)	59.7 (785)	468.4 (6182)	192.1 (2536)	141.6 (1869)	48.3 (638)	152.1 (2007)	162.0 (2149)	282.6 (3730)	185.0 (2442)	263.3 (3475)
	2) Minimum: Litres/sec. (IGPM)	28.3 (374)	31.4 (414)	31.4 (414)	48.3 (637)	25.3 (334)	22.4 (296)	7.8 (103)	9.9 (131)	25.3 (334)	51.9 (685.1)	48.3 (637)	44.7 (590)
	3) Average: Litres/sec. (IGPM)	44.0 (581)	39.6 (523)	40.1 (527)	200.6 (2648)	69.0 (911)	64.5 (851)	21.5 (284)	42.8 (565)	74.3 (981)	101.8 (1343.8)	79.8 (1053)	108.6 (1433)
	Total Volume Treated: Litres x 10^6 (Imp. Ga. x 10^6)	117.850 (25.923)	95.800 (21.073)	107.404 (23.626)	519.955 (114.374)	184.810 (40.667)	167.184 (36.775)	57.586 (12.667)	114.636 (25.216)	192.585 (42.362)	272.661 (59.977)	206.842 (45.499)	290.874 (63.983)
LIME	pH: 1) Feed ____ N-17	5.15	5.31	4.79	3.25	3.52	4.05	3.58	3.96	2.85	3.14	3.58	3.12
	2) Discharge ____ N-18	9.88	10.03	9.30	8.78	9.56	10.29	8.35	9.86	9.14	9.58	8.40	8.47
	Acidity, mg/L as $CaCO_3$ 1) Feed ____ N-17	504	515	540	328	510	724	706	836	620	645	539	565
	Lime Consumption 1) Tonnes CaO (Metric) (Short Imp. Tons CaO)	54.9 (60.5)	51.0 (56.2)	62.7 (69.1)	168.6 (185.8)	95.1 (104.8)	51.0 (56.2)	37.2 (41.0)	49.0 (54.0)	96.0 (106.0)	131.3 (144.7)	105.8 (116.6)	129.4 (142.6)
	2) mg CaO/litre (lbs. CaO/1,000 Imp.)	465.8 (4.68)	532.4 (5.33)	583.8 (5.85)	324.3 (3.25)	514.6 (5.15)	305.1 (3.06)	646.0 (6.47)	427.4 (4.28)	498.5 (5.00)	481.6 (4.83)	511.5 (5.13)	444.9 (4.46)
BARIUM CHLORIDE	**Dissolved Radium, pc/Litre** 1) Feed ____ N-17	5.4	5.3	5.9	17.1	14.0	10.0	8.0	8.0	12.0	8.0	8.0	8.0
	2) Discharge ____												
	Barium Chloride Consumption 1) Kilograms (pounds)	1550 (3410)	1350 (2970)	1500 (3300)	3700 (8140)	2450 (5390)	1300 (2860)	800 (1760)	1300 (2860)	1850 (4070)	3250 (7150)	2400 (5280)	3100 (6820)
	2) mg/l (lbs./1,000 Imp. Gal)	13.2 (0.13)	14.1 (0.141)	14.0 (0.14)	7.12 (0.071)	13.3 (0.133)	7.78 (0.078)	13.9 (0.140)	11.3 (0.11)	9.6 (0.096)	11.9 (0.12)	11.6 (0.116)	10.7 (0.107)
	Effluent at Property Boundary: N-19 1) pH	8.49	8,66	8.66	8.55	8.94	8.15	7.57	7.30	7.52	7.45	7.51	7.21
	2) Total Radium, pc/Litre	1.7	1.0	2.2	8.1	2.9	1.9	1.0	1.0	1.0	1.0	4.0	2.0
	Rainfall: 1) millimetres (inches)												

C-93

1982 EFFLUENT TREATMENT PLANT SUMMARY FOR CROTCH LAKE

ITEM		JAN.	FEB.	MAR.	APR.	MAY	JUN.	JUL.	AUG.	SEP.	OCT.	NOV.	DEC
EFFLUENT FLOW RATE	Days Plant Operated This Month:	31	28	31	30	31	30	31	31	30	31	30	31
	1) Maximum: Litres/sec. (IGPM)	111.5 (1472)	111.5 (1472)	115.7 (1527)	925.6 (12216)	797.6 (10528)	119.9 (1582)	115.7 (1527)	128.3 (1693)	217.6 (2872)	407.6 (5380)	418.6 (5525)	173. (2889
	2) Minimum: Litres/sec. (IGPM)	0.1 (1)	0.1 (1)	0.1 (1)	46.9 (619)	1.6 (21)	0 (0)	0.1 (1)	0 (0)	13.4 (177)	7.9 (104)	0.2 (3)	38.0 (502
	3) Average: Litres/sec. (IGPM)	50.9 (672)	54.6 (721)	38.4 (507)	473.7 (6252)	188.1 (2483)	22.7 (300)	47.7 (630)	37.3 (492)	119.5 (1577)	196.8 (2598)	95.1 (1255)	81.2 (1072
	Total Volume Treated: Litres x 10^6 (Imp. Ga. x 10^6)	136.331 (29.989)	132.088 (29.055)	102.851 (22.634)	1227.83 (270.085)	503.807 (110.841)	58.838 (12.943)	127.760 (28.103)	99.904 (21.976)	309.744 (68.134)	527.109 (115.948)	246.499 (54.222)	217.4 (47.84
LIME	pH: 1) Feed ________ CL-1	5.07	6.08	6.27	3.72	4.31	4.10	4.30	6.10	6.43	3.66	2.74	2.74
	2) Discharge ________ CL-2	11.94	11.33	11.99	10.80	11.28	11.78	10.34	9.25	9.41	10.35	11.1	10.3
	Acidity, mg/L as $CaCO_3$ 1) Feed ________ CL-1	483	133	289	170	208	306	358	853	140	318	353	349
	Lime Consumption 1) Tonnes CaO (Metric) (Short Imp. Tons CaO)	60.8 (67.0)	58.8 (64.8)	66.6 (73.4)	141.1 (155.5)	100.9 (111.2)	43.1 (47.5)	35.3 (38.9)	19.6 (21.6)	68.6 (75.6)	86.2 (95.0)	68.6 (75.6)	76. (84.
	2) mg CaO/litre (lbs. CaO/1,000 Imp.)	446.0 (4.47)	445.2 (4.46)	647.5 (6.49)	114.9 (1.15)	200.3 (2.01)	732.5 (7.34)	276.3 (2.77)	196.2 (1.97)	221.5 (2.22)	163.5 (1.64)	278.3 (2.79)	351. (3.5
BARIUM CHLORIDE	**Dissolved Radium, pc/Litre** 1) Feed ________												
	2) Discharge ________												
	Barium Chloride Consumption 1) Kilograms (pounds)	1500 (3300)	1400 (3080)	1600 (3520)	3750 (8250)	2500 (5500)	1250 (2750)	800 (1760)	500 (1100)	1900 (4180)	2200 (4840)	1850 (4070)	175 (385
	2) mg/l (lbs./1,000 Imp. Gal)	11.0 (0.11)	10.6 (0.106)	15.6 (0.156)	3.05 (0.031)	5.0 (0.050)	21.2 (0.212)	6.26 (0.063)	5.0 (0.05)	6.13 (0.06)	4.17 (0.04)	7.51 (0.075)	8.0 (0.08
Effluent at Property Boundary: CL-4	1) pH	7.20	6.70	7.00	6.8	8.85	8.95	8.15	7.9	7.3	7.3	7.10	6.8
	2) Total Radium, pc/Litre	12.0	9.8	15.0	8.6	8.5	10.0	9.0	10.0	12.0	16.0	8.0	7.0
Rainfall:	1) millimetres (inches)												

1982 EFFLUENT TREATMENT PLANT SUMMARY FOR PANEL

	ITEM	JAN.	FEB.	MAR.	APR.	MAY	JUN.	JUL.	AUG.	SEP.	OCT.	NOV.	DEC.
EFFLUENT FLOW RATE	Days Plant Operated This Month:	31	25	30	28	31	30	31	31	29	31	30	31
	1) Maximum: Litres/sec. (IGPM)	77.3 (1020)	79.6 (1051)	95.0 (1254)	163.5 (2158)	170.1 (2245)	143.6 (1895)	101.6 (1341)	59.6 (787)	99.5 (1313)	128.1 (1691)	126.4 (1668)	132.5 (1749)
	2) Minimum: Litres/sec. (IGPM)	72.9 (962)	35.3 (466)	19.3 (255)	75.2 (993)	143.6 (1896)	101.6 (1341)	61.8 (816)	37.6 (496)	66.3 (875)	99.4 (1312)	117.5 (1551)	121.5 (1604)
	3) Average: Litres/sec. (IGPM)	76.1 (1004)	70.4 (930)	82.2 (1085)	127.5 (1683)	160.1 (2113)	121.5 (1604)	86.4 (1140)	48.2 (636)	80.7 (1065)	112.7 (1488)	122.4 (1615)	126.2 (1666)
	Total Volume Treated: Litres x 10^6 (Imp. Ga. x 10^6)	203.826 (44.836)	152.064 (33.449)	213.062 (46.867)	308.448 (67.849)	428.812 (94.324)	314.928 (69.274)	231.414 (50.904)	129.099 (28.398)	202.202 (44.478)	301.856 (66.399)	317.261 (69.788)	338.014 (74.353)
LIME	pH: 1) Feed P-13	4.56	5.08	4.99	5.21	5.62	5.79	6.07	5.88	5.30	4.98	5.14	4.91
	2) Discharge P-14	8.81	6.89	9.02	8.86	9.18	10.17	9.55	9.46	9.05	7.98	9.06	9.22
	Acidity, mg/L as $CaCO_3$ 1) Feed P-13	24	20	20	16	9	5	4	7	14	18	32	17
	Lime Consumption 1) Tonnes CaO (Metric) (Short Imp. Tons CaO)	3.9 (4.3)	2.9 (3.2)	5.9 (6.5)	7.8 (8.6)	11.8 (13.0)	8.8 (9.7)	3.9 (4.3)	3.9 (4.3)	2.9 (3.2)	3.9 (4.3)	4.9 (5.4)	5.9 (6.5)
	2) mg CaO/litre (lbs. CaO/1,000 Imp.)	19.1 (0.19)	19.1 (0.19)	27.7 (0.28)	25.3 (0.254)	27.5 (0.276)	27.9 (0.280)	16.9 (0.169)	30.2 (0.30)	14.3 (0.144)	12.9 (0.13)	15.4 (0.15)	17.5 (0.17)
RADIUM CONTROL	Dissolved Radium, pc/Litre 1) Feed P-13	232	211	244	270	205	191	196	138	140	181	176	179
	2) Discharge P-14	3.4	2.8	4.4	2.9	3.5	2.1	1.8	2.4	2.3	3.2	6.2	3.2
	Barium Chloride Consumption 1) Kilograms (pounds)	3000 (6600)	1500 (3300)	3550 (7810)	5250 (11550)	5500 (12100)	6100 (13420)	4050 (8910)	2250 (4950)	4100 (9020)	4950 (10890)	3350 (7370)	3000 (6600)
	2) mg/l (lbs./1,000 Imp. Gal)	14.7 (0.147)	9.86 (0.099)	16.7 (0.17)	17.0 (0.170)	12.8 (0.128)	19.4 (0.194)	17.5 (0.175)	17.4 (0.174)	20.3 (0.203)	16.4 (0.16)	10.6 (0.106)	8.88 (0.089)
Effluent at Property Boundary: P-12	1) pH	7.55	6.70	8.50	7.55	7.60	8.65	7.76	7.25	7.20	6.90	7.15	7.40
	2) Total Radium, pc/Litre	12.0	7.4	16.0	11.5	8.5	4.3	2.0	1.0	5.0	1.0	9.0	6.0
Rainfall:	1) millimetres (Inches)												

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1982 EFFLUENT TREATMENT PLANT SUMMARY FOR QUIRKE

	ITEM	JAN.	FEB.	MAR.	APR.	MAY	JUN.	JUL.	AUG.	SEP.	OCT.	NOV.	DEC.
EFFLUENT FLOW RATE	Days Plant Operated this Month:	31	28	31	30	31	30	31	31	30	31	30	31
	1) Maximum: Litres/sec. (IGPM)	280.8 (3706)	261.9 (3457)	312.1 (4120)	468.9 (6189)	466.8 (6162)	299.6 (3954)	255.7 (3375)	199.3 (2630)	331.0 (4369)	500.3 (6604)	456.4 (6024)	393.6 (5195)
	2) Minimum: Litres/sec. (IGPM)	255.7 (3374)	230.6 (3043)	155.3 (2050)	331.0 (4369)	287.0 (3788)	29.9 (395)	130.3 (1720)	53.0 (700)	155.3 (2050)	337.2 (4451)	318.5 (4204)	324.7 (4285)
	3) Average: Litres/sec. (IGPM)	264.2 (3487)	245.4 (3239)	242.3 (3198)	431.1 (5690)	381.1 (5031)	(150.1) (1981)	188.9 (2493)	110.7 (1461)	265.1 (3499)	427.2 (5639)	379.4 (5007)	343.2 (4533)
	Total Volume Treated: Litres x 10^6 (Imp. Ga. x 10^6)	707.633 (155.756)	593.671 (130.590)	648.976 (142.755)	1117.41 (245.796)	1020.74 (224.584)	389.059 (85.581)	505.950 (111.293)	296.499 (65.221)	687.139 (151.149)	1144.212 (251.692)	983.405 (216.319)	919.227 (202.202)
LIME	pH: 1) Feed ______ 0-5	9.06	8.83	8.43	7.42	8.39	6.57	3.88	3.87	3.61	5.07	7.19	4.96
	2) Discharge ______ 0-3	8.75	8.25	8.08	7.94	7.80	8.18	7.40	8.61	7.00	8.66	8.77	8.59
	Acidity, mg/L as $CaCO_3$ 1) Feed ______ 0-5	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	77	107	159	81	22	59
	Lime Consumption 1) Tonnes CaO (Metric) (Short Imp. Tons CaO)	--	--	11.8 (13.0)	35.3 (38.9)	27.5 (30.3)	21.6 (23.8)	47.0 (51.8)	45.1 (49.7)	45.1 (49.7)	90.2 (99.4)	56.9 (62.8)	62.7 (69.1)
	2) mg CaO/litre (lbs. CaO/1,000 Imp.)	--	--	18.2 (0.18)	31.6 (0.317)	26.9 (0.270)	55.5 (0.556)	92.9 (0.931)	152.1 (1.52)	65.6 (0.66)	76.8 (0.79)	57.9 (0.58)	68.4 (0.68)
BARIUM CHLORIDE	**Dissolved Radium, pc/Litre** 1) Feed ______ 0-5	714	741	745	572	628	403	223	255	257	488	353	191
	2) Discharge ______ 0-3	8.6	4.5	3.5	3.6	2.9	1.5	2.0	1.2	3.5	3.4	3.7	1.8
	Barium Chloride Consumption 1) Kilograms (pounds)	7750 (17050)	7000 (15400)	7500 (16500)	12600 (27720)	12550 (27610)	5200 (11440)	5800 (12760)	4250 (9350)	7550 (16610)	12850 (28270)	7900 (17380)	8500 (18700)
	2) mg/l (lbs./1,000 Imp. Gal)	10.95 (0.110)	11.79 (0.12)	11.6 (0.12)	11.3 (0.11)	12.3 (0.123)	13.4 (0.134)	11.5 (0.115)	14.3 (0.14)	11.0 (0.11)	11.2 (0.11)	8.03 (0.080)	9.25 (0.093)
	Effluent at Property Boundary: 0-6 1) pH	8.45	7.65	7.90	7.65	6.55	7.35	5.00	8.25	5.40	8.05	8.80	8.95
	2) Diss. Radium, pc/Litre	3.1	2.5	1.8	3.4	3.9	4.1	2.0	1.0	2.0	3.0	3.0	1.0
	Rainfall: 1) millimetres (inches)												

1983 EFFLUENT TREATMENT PLANT SUMMARY FOR PRONTO

ITEM		JAN.	FEB.	MAR.	APR.	MAY	JUN.	JUL.	AUG.	SEP.	OCT.	NOV.	DEC.
EFFLUENT FLOW RATE	Days Plant Operated This Month:	0	0	31	30	31	30	31	31	20	0	30	9
	1) Maximum: Litres/sec. (IGPM)			204.9 (2704)	139.8 (1845)	64.2 (847)	59.6 (787)	69.0 (911)	89.4 (1180)	15.2 (201)		112.1 (1480)	112.1 (1480)
	2) Minimum: Litres/sec. (IGPM)			0 (0)	64.2 (847)	47.0 (620)	31.0 (409)	7.2 (95)	9.9 (131)	1.0 (13)		50.0 (660)	1.9 (25)
	3) Average: Litres/sec. (IGPM)			135.3 (1786)	114.0 (1505)	54.0 (713)	44.1 (582)	32.6 (430)	55.5 (733)	7.2 (95)		109.5 (1445)	83.5 (1102)
	Total Volume Treated: Litres x 10^6 (Imp. Ga. x 10^6)			362.388 (79.714)	295.488 (64.998)	144.634 (31.815)	114.307 (25.144)	87.316 (19.207)	148.651 (32.699)	12.442 (2.738)		283.824 (62.433)	64.930 (14.283)
LIME	pH: 1) Feed PR-2	3.83	4.80	6.28	3.64	3.36	3.18	3.02	2.95	3.08	3.54	3.47	4.18
	2) Discharge PR-4			9.54	9.38	10.73	10.68	10.40	10.02	8.82		9.85	9.76
	Acidity, mg/L as $CaCO_3$ 1) Feed PR-2	65	39	32	98	102	126	121	155	140	69	75	54
	Lime Consumption 1) Tonnes CaO (Metric) (Short Imp. Tons CaO)			39.2 (43.2)	29.4 (32.4)	21.6 (23.8)	21.6 (23.8)	17.6 (19.4)	19.6 (21.6)	1.0 (1.1)		23.5 (25.9)	2.9 (3.2)
	2) mg CaO/litre (lbs. CaO/1,000 Imp.)			108.2 (1.08)	99.5 (1.00)	149.3 (1.50)	189.0 (1.89)	201.6 (2.02)	132 (1.32)	80.4 (0.80)		82.8 (0.830)	44.7 (0.448)
BARIUM CHLORIDE	Total Radium pc/Litre 1) Feed PR-2		2	1	4	8	13	9	13	17		12	3
	2) Discharge PR-4			2	3	2	3	3	4	3		2	2
	Barium Chloride Consumption 1) Kilograms (pounds)			1100 (2420)	600 1320	650 (1430)	500 (1100)	500 (1100)	650 (1430)	250 (550)		1050 (2310)	100 (220)
	2) mg/l (lbs./1,000 Imp. Gal)			3.04 (0.03)	2.03 (0.020)	4.49 (0.045)	4.37 (0.044)	5.73 (0.057)	4.37 (0.044)	20.1 (0.20)		3.70 (0.037)	1.54 (0.015)
Effluent at Property Boundary: PR-1	1) pH	6.28	6.57	9.81	8.29	7.06	7.04	7.24	6.76	6.83	6.72	7.14	7.35
	2) Total Radium, pc/Litre		1	1	1	1	11	2	3	2		<1	1

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1983 EFFLUENT TREATMENT PLANT SUMMARY FOR NORDIC

	ITEM		JAN.	FEB.	MAR.	APR.	MAY	JUN.	JUL.	AUG.	SEP.	OCT.	NOV.	DEC.
EFFLUENT FLOW RATE	Days Plant Operated This Month:		31	28	31	30	31	30	31	31	30	31	30	31
	1) Maximum: Litres/sec. (IGPM)		120.0 (1584)	55.7 (735)	507.3 (6695)	699.7 (9235)	393.2 (5190)	357.4 (4717)	28.3 (374)	37.8 (499)	71.5 (944)	168.3 (2221)	220.0 (2904)	84.3 (1113)
	2) Minimum: Litres/sec. (IGPM)		34.5 (455)	41.2 (544)	41.2 (544)	51.9 (685)	44.7 (590)	19.7 (260)	17.0 (224)	17.0 (244)	25.3 (344)	31.4 (414)	31.4 (414)	34.5 (455)
	3) Average: Litres/sec. (IGPM)		62.6 (826)	44.5 (587)	119.4 (1576)	181.7 (2398)	83.9 (1107)	65.2 (861)	22.0 (290)	28.0 (370)	32.2 (425)	55.0 (726)	69.7 (920)	45.3 (598)
	Total Volume Treated: Litres x 10^8 (Imp. Ga. x 10^6)		167.668 (36.882)	107.654 (23.681)	319.801 (70.346)	470.966 (103.598)	224.718 (49.431)	168.990 (37.174)	58.925 (12.962)	74.995 (16.497)	83.482 (18.359)	147.312 (32.404)	180.662 (39.740)	117.418 (25.828)
LIME	pH: 1) Feed	N-17	3.70	4.83	3.61	3.70	3.38	4.11	3.83	4.92	4.32	3.30	3.48	3.77
	2) Discharge	N-18	9.50	9.14	8.46	8.19	8.21	9.12	9.83	10.65	11.27	9.85	8.73	8.85
	Acidity, mg/L as $CaCO_3$ 1) Feed	N-17	685	808	533	356	631	1084	1083	1100	1089	915	877	879
	Lime Consumption 1) Tonnes CaO (Metric) (Short Imp. Tons CaO)		96.0 (105.8)	54.9 (60.5)	134.2 (147.9)	147.0 (162.0)	96 (105.8)	86.2 (95.0)	51.0 (56.2)	51.0 (56.2)	56.8 (62.6)	84.3 (92.9)	84.3 (92.3)	66.6 (73.4)
	2) mg CaO/litre (lbs. CaO/1,000 Imp.)		572.6 (5.74)	510.0 (5.11)	419.6 (4.20)	312.1 (3.13)	427.2 (4.28)	510.1 (5.11)	865.5 (8.67)	680.0 (6.81)	680.5 (6.82)	572.3 (5.73)	466.6 (4.65)	567.2 (5.68)
BARIUM CHLORIDE	Diss. Radium, pc/Litre 1) Feed	N-17	6	4	9	10	13	8	9	10	9	12	10	9
	2) Discharge													
	Barium Chloride Consumption 1) Kilograms (pounds)		2550 (5610)	1350 (2970)	2600 (5720)	0 (0)	0 (0)	0 (0)	0 (0)	0 (0)	0 (0)	0 (0)	0 (0)	0 (0)
	2) mg/l (lbs./1,000 Imp. Gal)		15.21 (0.152)	12.54 (0.125)	8.13 (0.081)	0 (0)	0 (0)	0 (0)	0 (0)	0 (0)	0 (0)	0 (0)	0 (0)	0 (0)
	Effluent at Property Boundary: 1) pH	N-19	6.94	8.05	7.33	8.25	7.78	6.29	7.25	6.92	7.05	6.78	6.90	7.00
	2) Total Radium, pc/litre		3	2	2	10	4	8	4	5	3	4	3	3

1983 EFFLUENT TREATMENT PLANT SUMMARY FOR CROTCH LAKE

	ITEM	JAN.	FEB.	MAR.	APR.	MAY	JUN.	JUL.	AUG.	SEP.	OCT.	NOV.	DEC.
EFFLUENT FLOW RATE	Days Plant Operated This Month:	31	28	31	30	31	30	31	31	31	31	30	31
	1) Maximum: Litres/sec. (IGPM)	128.3 (1693)	50.0 (660)	222.7 (2999)	798.8 (10543)	298.4 (3939)	298.4 (3938)	128.3 (1693)	124.0 (1637)	91.8 (1212)	468.0 (6177)	333.1 (4397)	182.9 (2414)
	2) Minimum: Litres/sec. (IGPM)	29.8 (393)	29.8 (393)	40.9 (551)	19.9 (263)	50.0 (660)	84.3 (1113)	56.4 (745)	29.8 (393)	50.0 (660)	88.0 (1161)	119.9 (1583)	115.7 (1527)
	3) Average: Litres/sec. (IGPM)	59.3 (783)	40.3 (532)	94.2 (1268)	201.3 (2657)	123.3 (1628)	120.3 (1588)	89.0 (1175)	58.9 (777)	67.5 (891)	241.5 (3187)	180.5 (2383)	132.1 (1744)
	Total Volume Treated: Litres x 10^6 (Imp. Ga. x 10^6)	158.829 (34.937)	97.494 (21.446)	252.305 (55.499)	521.770 (114.773)	330.247 (72.644)	311.818 (68.590)	238.378 (52.436)	157.756 (34.702)	174.980 (38.486)	646.834 (142.284)	467.856 (102.914)	353.817 (77.829)
LIME	pH: 1) Feed ______ CL-1	2.89	3.06	3.14	3.21	2.91	3.60	4.13	6.85	5.46	6.38	8.45	11.11
	2) Discharge ______ CL-2	11.40	11.80	10.89	9.38	9.56	9.89	10.69	10.19	5.76			
	Acidity, mg/L as $CaCO_3$ 1) Feed ______ CL-1	324	438	279	354	374	210	100	86	52	39	15	
	Lime Consumption 1) Tonnes CaO (Metric) (Short Imp. Tons CaO)	64.7 (71.3)	51.0 (56.2)	82.3 (90.7)	86.2 (95.0)	92.1 (101.5)	74.5 (82.1)	35.3 (38.9)	3.9 (4.3)	0 (0)	0 (0)	0 (0)	0 (0)
	2) mg CaO/litre (lbs. CaO/1,000 Imp.)	407.4 (4.08)	523.1 (5.24)	326.2 (3.27)	165.2 (1.66)	278.0 (2.79)	238.9 (2.39)	148.1 (1.484)	24.7 (0.248)	0 (0)	0 (0)	0 (0)	0 (0)
BARIUM CHLORIDE	Radium, pc/litre 1) Feed ______ CL-1						166	429	382	381	435	418	408
	2) Discharge ______												
	Barium Chloride Consumption 1) Kilograms (pounds)	1600 (3520)	1350 (2970)	1900 (4180)	1900 (4180)	2000 (4400)	1800 (3960)	1350 (2970)	1950 (4290)	4200 (9240)	3600 (7920)	4850 (10670)	4900 (10780)
	2) mg/l (lbs./1,000 Imp. Gal)	10.07 (0.101)	13.85 (0.139)	7.53 (0.075)	3.64 (0.036)	6.06 (0.061)	5.77 (0.058)	5.66 (0.057)	12.36 (0.124)	24.0 (0.24)	5.57 (0.06)	10.37 (0.104)	13.85 (0.139)
Effluent at Property Boundary: CL-4	1) pH	7.0	6.20	6.60	6.35	6.30	6.74	7.49	7.33	7.02	6.17	5.89	5.55
	2) Total Radium, pc/Litre	14	7	11	12	12	11	15					
	3) Diss. Radium, pc/Litre								13	9	8	5	3

1983 EFFLUENT TREATMENT PLANT SUMMARY FOR PANEL

	ITEM		JAN.	FEB.	MAR.	APR.	MAY	JUN.	JUL.	AUG.	SEP.	OCT.	NOV.	DEC.
EFFLUENT FLOW RATE	Days Plant Operated This Month:		31	26	31	30	31	30	31	31	30	31	30	31
	1) Maximum: Litres/sec. (IGPM)		132.6 (1750)	145.9 (1926)	114.9 (1516)	161.2 (2128)	159.0 (2099)	154.6 (2040)	121.5 (160.4)	66.3 (875)	72.9 (962)	94.9 (1253)	110.5 (1459)	110.5 (1459)
	2) Minimum: Litres/sec. (IGPM)		121.5 (1604)	127.3 (1680)	106.0 (1399)	83.5 (1102)	119.7 (1580)	121.5 (1604)	22.1 (292)	0 (0)	0 (0)	70.7 (933)	0 (0)	0 (0)
	3) Average: Litres/sec. (IGPM)		125.9 (1662)	139.4 (1840)	110.5 (1458)	122.4 (1615)	149.9 (1979)	142.0 (1874)	90.8 (1199)	56.6 (747)	52.6 (694)	82.9 (1094)	75.2 (993)	99.2 (1309)
	Total Volume Treated: Litres x 10^6 (Imp. Ga. x 10^6)		337.211 (74.176)	313.148 (68.883)	295.963 (65.103)	317.261 (69.788)	401.492 (88.316)	368.064 (80.963)	243.190 (53.496)	151.597 (33.347)	136.339 (29.990)	222.039 (48.842)	194.918 (42.876)	265.697 (58.445)
LIME	pH: 1) Feed	P-13	4.96	5.20	5.38	5.31	5.42	5.33	5.61	5.61	5.60	5.04	5.46	4.83
	2) Discharge	P-14	8.68	8.85	8.98	8.92	9.25	8.98	9.38	9.81	9.59	9.64	9.54	9.15
	Acidity, mg/L as $CaCO_3$ 1) Feed	P-13	16	18	20	18	12	14	11	9	11	16	19	20
	Lime Consumption 1) Tonnes CaO (Metric) (Short Imp. Tons CaO)		5.9 (6.5)	4.9 (5.4)	6.9 (7.6)	6.9 (7.6)	8.8 (9.7)	5.9 (6.5)	7.8 (8.6)	4.9 (5.4)	5.9 (6.5)	6.9 (7.6)	5.9 (6.5)	4.9 (5.4)
	2) mg CaO/litre (lbs. CaO/1,000 Imp.)		17.50 (0.175)	15.65 (0.157)	23.3 (0.234)	21.7 (0.218)	21.9 (0.220)	16.0 (0.16)	32.1 (0.322)	32.3 (0.324)	43.3 (0.433)	31.1 (0.311)	30.27 (0.303)	18.44 (0.185)
BARIUM CHLORIDE	**Total Radium, pc/Litre** 1) Feed	P-13	174	201	193	198	152	176	177	139	174	184	171	222
	2) Discharge	P-14	12	7	5	4	4	5	7	13	8	6	9	11
	Barium Chloride Consumption 1) Kilograms (pounds)		3550 (7810)	2750 (6050)	3250 (7150)	3550 (7810)	4100 (9020)	3550 (7810)	2300 (5056)	1050 (2310)	1500 (3300)	2200 (4840)	2400 (5280)	3850 (8470)
	2) mg/l (lbs./1,000 Imp. Gal)		10.53 (0.105)	8.78 (0.088)	10.9 (0.110)	11.2 (0.112)	10.2 (0.102)	9.6 (0.097)	9.46 (0.095)	6.93 (0.069)	11.0 (0.110)	9.91 (0.099)	12.3 (0.123)	14.5 (0.145)
Effluent at Property Boundary:	1) pH													
	2) Radium pc/litre													

1983 EFFLUENT TREATMENT PLANT SUMMARY FOR QUIRKE

	ITEM	JAN.	FEB.	MAR.	APR.	MAY	JUN.	JUL.	AUG.	SEP.	OCT.	NOV.	DEC.
EFFLUENT FLOW RATE	Days Plant Operated This Month:	31	28	31	30	31	30	31	31	30	31	30	31
	1) Maximum: Litres/sec. (IGPM)	393.6 (5195)	324.7 (4286)	287.0 (3788)	487.7 (6437)	544.2 (7183)	437.5 (5774)	268.2 (3540)	236.9 (3127)	230.6 (3044)	243.1 (3208)	218.1 (2879)	155.4 (2051)
	2) Minimum: Litres/sec. (IGPM)	268.2 (3540)	280.8 (3707)	249.7 (3296)	274.5 (3623)	374.8 (4947)	249.4 (3292)	224.3 (2960)	199.2 (2629)	205.5 (2713)	211.8 (2795)	28.1 (371)	33.1 (437)
	3) Average: Litres/sec. (IGPM)	336.6 (4443)	310.6 (4100)	266.5 (3517)	352.3 (4650)	451.7 (5962)	375.7 (4959)	248.9 (3285)	208.6 (2754)	217.4 (2870)	222.4 (2935)	70.6 (932)	117.8 (1555)
	Total Volume Treated: Litres x 10^6 (Imp. Ga. x 10^6)	901.549 (198.313)	751.404 (165.286)	713.794 (157.013)	913.162 (200.867)	1209.833 (266.126)	973.814 (214.209)	666.654 (146.643)	558.714 (122.900)	563.501 (123.953)	595.676 (131.030)	182.995 (40.253)	315.516 (69.404)
LIME	pH: 1) Feed Q-5	5.86	6.57	5.69	5.04	3.71	3.98	3.99	3.65	3.70	3.51	3.26	3.19
	2) Discharge Q-3	8.85	8.14	7.97	7.95	7.22	8.50	7.77	7.70	8.50	8.49	8.20	9.56
	Acidity, mg/L as $CaCO_3$ 1) Feed Q-5	35	29	59	45	106	66	44	93	93	126	123	197
	Lime Consumption 1) Tonnes CaO (Metric) (Short Imp. Tons CaO)	56.8 (62.6)	45.1 (49.7)	56.8 (62.6)	64.7 (71.3)	84.3 (92.9)	68.6 (75.6)	52.9 (58.3)	52.9 (58.3)	60.8 (67.0)	64.7 (71.3)	45.1 (49.7)	58.8 (64.8)
	2) mg CaO/litre (lbs. CaO/1,000 Imp.)	63.0 (0.631)	60.02 (0.601)	79.6 (0.797)	70.9 (0.710)	69.8 (0.698)	70.4 (0.706)	79.4 (0.795)	94.7 (0.949)	107.9 (1.08)	108.6 (1.09)	246.5 (2.46)	186.4 (1.867)
BARIUM CHLORIDE	Total Radium, pc/Litre 1) Feed Q-5	339	381	273	218	242	227	55	25	26	60	123	143
	2) Discharge Q-3	24	58	16	12	37	12	4	2	2	3	5	5
	Barium Chloride Consumption 1) Kilograms (pounds)	8500 (18700)	10950 (24090)	9850 (21670)	12500 (27500)	10400 (22880)	11150 (24530)	10750 (23650)	8000 (17600)	4650 (10230)	3750 (8250)	3450 (7590)	4600 (10120)
	2) mg/l (lbs./1,000 Imp. Gal)	9.43 (0.094)	14.57 (0.146)	13.8 (0.138)	13.9 (0.137)	8.60 (0.086)	11.45 (0.115)	16.13 (0.161)	14.3 (0.143)	8.25 (0.083)	6.30 (0.063)	18.85 (0.189)	14.58 (0.146)
	Effluent at Property Boundary: Q-6 1) pH	6.70	6.15	9.10	7.76	5.85	8.90	5.19	6.40	8.06	8.55	8.70	6.60
	2) Diss. Radium, pc/Litre	2	1	1	<1	1	11	1	1	1	1	<1	2
	3) Total Radium, pc/Litre	7		3		2	12	2		1		1	

C101

1983 EFFLUENT TREATMENT PLANT SUMMARY FOR STANLEIGH

	ITEM	JAN.	FEB.	MAR.	APR.	MAY	JUN.	JUL.	AUG.	SEP.	OCT.	NOV.	DEC.
EFFLUENT FLOW RATE	Days Plant Operated This Month:												20
	1) Maximum: Litres/sec. (IGPM)												599.6 (7915)
	2) Minimum: Litres/sec. (IGPM)												52.6 (694)
	3) Average: Litres/sec. (IGPM)												450.1 (5941)
	Total Volume Treated: Litres x 10^6 (Imp. Ga. x 10^6)								26.508 (5.831)	49.230 (10.829)	404.706 (89.023)	410.314 (90.256)	777.773 (171.086)
LIME	pH: 1) Feed ____ CL-4								7.33	7.02	6.17	5.69	5.55
	2) Discharge ____ CL-6									7.43	6.08	6.74	6.90
	Acidity, mg/L as $CaCO_3$ 1) Feed ____ CL-4								3	3	5	8	21
	Lime Consumption 1) Tonnes CaO (Metric) (Short Imp. Tons CaO)								1.2 (1.3)	4.5 (5.0)	2.0 (2.2)	1.9 (7.1)	2.0 (2.1)
	2) mg CaO/litre (lbs. CaO/1,000 Imp.)								45.3 (0.45)	91.4 (0.92)	4.9 (0.05)	4.63 (0.047)	2.57 (0.025)
BARIUM CHLORIDE	Radium, pc/litre (Dissolved) 1) Feed ____ CL-4								13	9	6	5	3
	2) Discharge ____ CL-6									2	1	3	1
	Barium Chloride Consumption 1) Kilograms (pounds)								907 (2000)	907 (2000)	3720 (8200)	3000 (6600)	900 (1980)
	2) mg/l (lbs./1,000 Imp. Gal)								34.2 (0.34)	18.4 (0.18)	9.19 (0.09)	7.31 (0.073)	1.16 (0.012)
Effluent at Property Boundary:	1) pH												
	2) Radium pc/litre												

APPENDIX D

NOTES FROM RON NICHOLSON

This Appendix contains a statement from Ron Nicholson, a doctorate student at the University of Waterloo, who is doing work on pyrite oxidation kinetics. His particular research is focussed on the effect of carbonate ion on the rate of pyrite oxidation. His notes refer to the problem of control of the oxidation rate of pyrite flotation tailings by the addition of calcite. The data are referred to in Section 5.4, and Section 7.2.

The notes sent by Mr. Nicholson are typed. The method of calculation which he employed is also included. Thanks are extended to the staff of the National Uranium Tailings program for the typing, which included many symbols not found on a conventional typewriter.

R. V. Nicholson
Institute for Groundwater Research
University of Waterloo
Waterloo, Ontario
N2L 3G1

April 1984

Mr. K. D. Hester
K. D. Hester and Associates
302 Pinehurst Drive
Oakville, Ontario
L6J 4X3

Dear Mr. Hester:

In regard to your letter dated March 28, 1984, I have performed some calculations related to the probable rate of limestone removal (dissolution) from a protective cover-layer should it be placed above a pyritic tailings deposit. As requested, the specifications of such a cover-layer were considered for the following conditions. The top cover would consist of a 2.3 m deep (approx.) layer of depyritized tailings containing 0.24% S (0.43% pyrite). This cover would also contain 12 kg/t of ground limestone (minus 44 microns with 95% available $CaCO_3$). An intermediate layer overlying the pre-deposited tailings and underlying the cover-layer consists of a 0.2 m (approx.) thick layer containing the pyrite float products at 29.0% S (54.3% pyrite).

For the purposes of these calculations, I have made some assumptions for properties of the cover as well as the available pyrite. The bulk density of the cover was set at 1600 kg/m^3, the average particle diameter of pyrite grains was estimated to be about 70 microns, giving a specific surface area of 17 m^2/kg and the temperature was assumed to be 10^oC. Although the calculations are generally straightforward concerning oxidation rates, many uncertainties exist with regard to the associated physiochemical behavior in a porous medium. For example, it is not well understood at which point in time a limestone-bearing pyritic-deposit will begin to turn acidic. Whether this time is associated with complete removal (dissolution) of $CaCO_3$ from a particular zone or throughout the complete layer is unknown. And again, although estimates of oxygen transport into the cover can be made, it is likely that the topmost zone does not behave ideally and would probably allow greater gas entry than would be attributed to the same material at depth. In view of these uncertainties, I have presented best-case/worst-case scenarios for this hypothetical problem.

The summary of depletion ($CaCO_3$) times attributed to different processes is given in the attached table after which are listed the pertinent assumptions and brief explanations.

Summary Table:

	Scenario			
	I	II	III	II + III
Process	Best Case Oxidation Only	Worst Case Oxidation Only	Natural Leaching Only	Worst Case Pyrite Oxidation Natural+Leaching
Depletion Time (years approx.)	230	75	60	30
*Assumptions Used	1), 2), 3)	2), 3), 4)	3), 5)	2), 3), 4), 5)

*1) Ideal diffusion conditions - Diffusive transport equation is applicable with pyrite oxidation as a sink for oxygen.

2) Rate of oxidation constant with time - The experimental rate-equation holds true for long periods of time.

3) Critical zone of 0.2 m depth only - Assume that when $CaCO_3$ is depleted in this zone (topmost layer) acidic conditions would be initiated. These conditions would encourage bacterial

growth and catalysis of the oxidation reaction would begin. Because the oxidation rates (bacteria) are many orders of magnitude greater than those in inorganically-controlled systems, the further acidification of the remaining deposit would proceed relatively fast.

4) Atmospheric levels of O_2 gas in the top 0.2 m zone of the cover - This is likely when the surface zone dries during evaporation in the warmer seasons.

5) Partial pressure of CO_2 gas slightly above atmospheric levels ($PCO_2=10^3$ atm). - If a natural soil profile develops, the CO_2 partial pressure will increase and the limestone will dissolve to a greater degree and thus be removed in a shorter time.

Conclusions:

(1) Acidic conditions in the topmost 20 cm of the cover-layer are anticipated after 30 to 60 years of emplacement.

(2) After $CaCO_3$ is completely removed from the cover, reduction of the flux of acid into the tailings will still be realized. The net acid input, however, will likely result in pH reduction due to lack of significant buffer material. The resulting pH is not known and further, more sophisticated modelling is needed to determine the resulting character of the pore-water chemistry.

(3) Acid rain input does not significantly affect the estimated rates of limestone dissolution.

Additional Remarks:

(1) The proposed quantity of limestone is not sufficient to consume the total potential acidity with the sulphur content specified. At least 15 kg/t of limestone is needed at the expected grade. ($CaCO_3$/Pyrite = 3.3 by weight).

(2) The effects of placing additional limestone over (tilled into?) the tailings at 25 (to 50?) year intervals is not known. Initial reports in the literature, however, suggest that this treatment does not guarantee the maintainance of neutral - pH pore-water at depth.

I hope that this material is of use to you. If there are any questions or comments please don't hesitate to call.

Yours Sincerely
R.V. Nicholson

Brief summary of calculations used to determine H^+ input rates and $CaCO_3$ dissolution rates in a potential cover on pyritic tailings.

Pyrite: Particle Diameter $d = 70\mu$ ($As = 1.7 \times 10^{-2}\ m^2 g^{-1}$)

(1) Worst Case - Atmosphere O_2 in Top 0.2 m layer.

Experimental Rate Equation

$$\frac{-dN_{O2}}{dt} = K'S\,C^n$$

; N_{O2} = moles of O_2 consum

t = time

$K' = 6.28 \times 10^{-7}\ m\ h^{-1}$ (experimental)

$C = 9.04\ mol\text{-}O_2\ m^{-3}$ (atmospheric O_2)

$$S = \underbrace{7.0\ kg\ m^{-3}}_{\text{Pyrite Content}} \times \underbrace{17\ m^2\ kg^{-1}}_{\text{As}} \times \underbrace{0.2\ m}_{\text{Layer thickness}} = 23.8\ m^2$$

$$\frac{-dNH^+}{dt} = \frac{16}{15}\frac{dN_{O2}}{dt} = 5.48 \times 10^{-5}\ mol\ h^{-1}$$

from dissolution reaction $1\ mol\text{-}H^+ + 1\ mol\ CaCO_3 \longrightarrow Ca^{2+} + HCO_3^-$

mass of $CaCO_3$ in 0.2 m zone = $182.4\ mol\ m^{-3} \times 0.2\ m^3 = \underline{36.48\ mol}$

Time for Dissolution $= \tau = M_{caco_3}/RH+$

$= 36.48\ mol / 5.48 \times 10^{-5}\ mol\ h^{-1}$

$= 6.66 \times 10^5\ h$

$= 76\ a$

Please note that numbers here for (1) and (2) should be reversed to conform to chart sent to you previously.

(2) Best Case Scenario

O_2 Diffusive Transport Controlled

$F_{O2} = \alpha C_o \sqrt{KD}\,[1-\alpha\sqrt{\frac{K}{D}}\;z]^{\gamma}$;

F_{O2} = flux of O_2 into media at z
z = depth from surface
C_o = atmospheric O_2 concentration
K = rate constant
D = diffusion coeff. (effective)
α = const = f(Co)
γ = const = f(n)
n = rate order for O_2 in rate equation

Flux difference across the 0.2 m zone gives the mass of O_2 consumed in this zone.

F_1 (at $z = 0$) = 3.55×10^{-5} mol m^{-2} h^{-1}
F_2 (at $z = 0.2$) = $3.55 \times 10^{-5} \times 0.72$ mol m^{-2} h^{-1}
$\Delta F = 9.9 \times 10^{-6}$ mol m^{-2} $h^{-1} \simeq \Delta F_{H^+} = \Delta F_{caCO_3}$
$T = M_{CaCO_3}/\Delta F_{H^+} = 36.48$ mol $m^{-2}/9.95 \times 10^{-6}$ mol m^{-2} h^{-1}
$= 3.67 \times 10^{6}$ h
= 420 a (This value differs from that sent originally because a different value for D and K were used here to give a better "BEST" case estimate)

(3) Calculation of Natural Dissolution Rates for $CaCO_3$

a) Equilibrium concentration of $CaCO_3$ in solution

Ionic strength = 0.01;∴ $\gamma_{HCO_3} = 0.9$, $\gamma_{Ca^{2+}} = 0.7$

at 10°C $K_0 = 10^{-1.27}$

$K_1 = 10^{-6.4}$

at pH ~ 6 to 8 no CO_3^{2-} is present (negligeable)

* ① $[H^+] = ([Ca^{2+}] PCO_2/10^{9.98})^{1/2}$

② $(Ca^{2+}) = (HCO_3^-)/2$

③ $(Ca^{2+}) = [Ca^{2+}]/\gamma Ca^{2+}$

④ $[HCO_3^-] = K.K.PCO_2/[H^+]$

⑤ $(HCO_3^-) = [HCO_3^-]/\gamma HCO_3$

* () is concentration units

[] is activities

Iterative solution for $[Ca^{2+}]$ and then (Ca^{2+}) gives at $PCO_2 = 10^{-3}$ (low)

$[Ca^{2+}] = 8.71 \times 10^{-4}$

$(Ca^{2+}) = 1.24 \times 10^{-3}$

$(HCO_3^-) = 2.49 \times 10^{-3}$ mol L^{-1}

b) Flux of $CaCO_3$ (dissolved in pore water) passing through cover layer (top 0.2 m)

$(CaCO_3) = 1.25$ mol L^{-1} - equilibrium concentration

Rate of water infiltration:

$F_{H_2O} = 500 \text{ L m}^{-2} \text{ a}^{-1}$

$FCaCO_3 = F_{H_2O} \times C_{CaCO_3}$

$= 0.625 \text{ mol m}^{-2} \text{ a}^{-1}$

$\tau = 36.48 \text{ mol m}^{-2} \text{ a}^{-1} \simeq 60 \text{ a}$

(4) Worst case oxidation plus natural dissolution

Fluxes additive

[illegible] $CO_3 = F$

$\longrightarrow$from dissolution

$\rightarrow$from pyrite oxidation

$F = 0.48 + 0.625 \text{ mol m}^{-2} \text{ a}^{-1} = 1.1 \text{ mol m}^{-2} \text{ a}^{-1}$

$\tau \simeq 30$ a

APPENDIX E

DISCUSSION NOTES.

Ron Nicholson - University of Waterloo, Dep't of Earth Sciences.

The oxidation of pyrite giving rise to large increases in acidity is probably uncontrollable once bacterial oxidation has started. In new tailings, the objective should be to prevent the pH from dropping to a level of about 4.5, at which point bacterial action kicks in. How to do this?

At a pH of 7.0 or so, there is still some appreciable oxidation rate attached to pyrite. This rate has been measured in the PYROX system of experiments. These addressed:

PYROX 1 - to determine surface treatment needed to obtain "reasonable" oxidation rates.

PYROX 2- to check effect of surface area on oxidation rates.

PYROX 3 - to check effect of oxygen concentration on oxidation rates.

PYROX 4 - to check the effect of temperature.

PYROX 5 - to check the difference between pyrites from differing locations.

It is not possible to prevent pyrite oxidation. The challenge is to keep it low enough so it isn't a problem. There will always be some permeation of solution through a tailings basin. The basin can be considered a large "basket". The active layer is the top, above the water table. Oxidation rates within the "dry" layer will depend on oxygen flux into this layer, and concentration throughout the layer. It will also depend on the "saturation" of this dry layer. A layer of tailings having considerable pore water will have a lower flux, more resistance, than a bone dry layer.

If there is an inert cover on the pyritic material, the oxygen flux down to the pyritic containing material will depend on the diffusivity of oxygen through this layer, and the thickness of the layer.

Dr. K. Carl Ivarson - Federal Dep't of Agriculture.

Excavations at a major Bell Canada building in Ottawa were made in a pyrite containing shale. Foundations were poured on supports immersed in the shale. Some time after the concrete covering had been installed, concrete movement and breakage was evident. The cause was the formation of acid and iron sulphates from the pyrite. The expansion pressures generated by the formation of these substances from the sulphides cracked the overlying concrete. In order to stop the continued heaving, KOH was used to neutralize the acidity. Of course, eventually the basic material escaped, leading to further problems. This occurrence, which has been written up in the literature, shows the ubiquitous and persistent nature of the thio bacillus.

Certain Canadian soils are derived from pyrite contining parent rock beneath. These "acid sulphate" soils are problems for agricultural use. Treatment of such soils involves "washing" the soil occasionally to remove sulphates and acidity. The addition of reducing materials (e. g. manure) slows sulphate formation or reverses it.

Jarosites are formed by the reaction of the sulphuric acid and iron salts formed from the constituents of the local rocks. These hydrated iron sulphate and alkali metal compounds are the end products of the biological oxidation of pyrite.

Sulphate reducing bacteria, acting on the jarosite, can reform the pyrite.

Because of the persistent nature, and the rapid reaction of the bacillus with pyrite, any vegetative cover approach will only work for a period. Intermittent addition of limestone may be needed to bring the pH back up to neutral, and prevent the acidification of the mass of pyrite. The limestone addition used to inhibit pyrite must be fine, since large lumps of limestone will not be effective.